PRE-BRAHM TESTS ON SULPHIDE ORES

1961363 - R8 SEMS

Responsive
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Kandal Meeting '88

To Jim Barron

HEAP LEACH TESTING

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Although performing a heap leach test is inherently simple, a lot of information beyond extraction, leach time, and reagent consumption can be obtained from a well designed test program. The following is a list of information to look for and acquire when performing column tests and field heap leach tests.

Ore

- define the ore types that will be encountered. Should these be combined or tested separately?
- perform mineralogy to define the minerals present and to define the manner in which values are associated with the minerals.
- perform repetitive assays on ore splits of various sizes to determine the best sampling procedure. This will be applicable to exploration drilling programs as well as the metallurgical programs. It will define the minimum representative sample size at various crush sizes to give an assay error that you can live with within a confidence limit that you specify.
- determine the crush size needed (extraction vs leach time vs cost).
- determine angle of repose of mined ore and crushed ore (stockpile sizing).
- bulk density of crushed ore
- crushability index
- abrasion index
- assays for alumina, silica, and iron (abrasion indicators)
- blend samples well before splitting.
- size distribution and assays by size fraction (ore, ore crushed to successively finer sizes), calculate head assays from screen assays.
- several splits for assayed heads by AA and fire assay
- moisture content (take with a grain of salt because it'll be drier than ore mined during commercial operations)

Agglomeration

- static strength tests
- dynamic strength tests
- amount of binder needed and type of binder
- water or a strong cyanide solution? Amount needed?
- add water as spray or droplets?
- mixing time needed
- type of agglomerator
- mix ore and binder dry before adding water.
- determine % moisture in fresh agglomerates.
- cure time needed (cover agglomerates while curing)
- let agglomerates cure in column.
- determine bulk density of fresh agglomerates for heap sizing.
- determine angle of repose of fresh agglomerates.
- observe the agglomerates through a plexiglass column for compaction, smearing, channeling, etc --- or, make these observations as the leached ore is slowly and carefully removed from the column with as little disturbance as possible.

Leaching

- weigh the ore being tested.
- make columns as high as possible up to the height of the expected commercial heap. If the ore has sulfides and this height of column is not practical, consider a salamander type column with sealed transfer points.
- determine dissolved oxygen in preg solutions before they have a chance to be re-aerated, particularly if there are significant oxygen consumers in the ore.
- load columns uniformly (turn columns, load thru center chute, etc).
- measure height of ore in column before and after leaching to determine slump.
- determine bulk density of ore in columns before and after leaching.
- apply leach solution at a uniform rate, i.e. peristaltic pumps rather than head tanks.
- perform tests at various flow rates, .002 to .01 gpm/ft^2.
- perform tests on ore crushed to various sizes.

- perform tests with and without a strong cyanide solution added during agglomeration.
- perform tests with the addition of surfactants to speed up leach rate.
- perform tests with the addition of oxygen to speed up leach rate.
- perform tests with and without agglomeration to determine effect on extraction, leach rate, and total suspended solids in preg solution.
- perform tests at various concentrations of lixiviant and note effect on extraction, leach rate, and reagent consumption.
- recycle preg solution to allow the buildup of impurities. Note the effect on leach rate and extraction; assay the saturated solution for permitting purposes.
- determine leach rates; plot these routinely as the tests progress.
- assay solutions as soon as possible for cyanide species; preserve the solutions with ascorbic acid.
- keep pH above 10.0 unless it is a variable being tested.
- in column tests, account for the volume of samples taken for assay and add these back to the metallurgical balance.
- keep the cyanide concentration constant during the leach test.
- add a means for uniformly distributing solution at the top of a column.
- determine the merits of spraying the column only 8 hours/day or only 16 hours/day so as to minimize the volume of the preg solution and the size of the recovery plant.
- determine the moisture content of drained ore after leaching (water balance).
- determine the volume of solution that will drain from a column when the sprays are shut off -- the preg ponds need to hold this volume along with other volume requirements. During this "draindown", determine a curve of volume drained vs time.
- for a reuseable leach pad project,
  - determine time from start of spraying to initial breakthrough, and to steady state preg flow.
  - determine the time needed to extract values, i.e., perform tests in columns as tall as the commercial heap or in a series of shorter columns to simulate commercial heap height.
  - determine the time needed to drain the column.
  - determine the time needed to detoxify the column.
  - determine detoxification procedures and costs.
  - determine a complete analysis of preg solution for attenuation studies.

- determine the soluble gold content of the final residues - is a water wash needed?
- determine the tendency of the residues to continue leaching after they are removed from the leach pad, i.e. EP Toxicity test or equivalent?

Solution handling

- pass the preg solution through a carbon column to remove the values before recycling the solution back to the column --- be sure that more than enough carbon is used to achieve low barrens.
- assay the barren solution for values and replenish cyanide and alkalinity if needed.
- get a complete analysis of the barren solution for permitting reasons.
- at the end of the leach test, reclaim the values from the carbon and compare the extraction so obtained with the extraction obtained from preg solution volumes and assays.
- determine the amount of mercury adsorbed on the carbon.
- assay the carbon for other adsorbed metals and back calculate the composition of saturated barren solution to simulate a Merrill-Crowe barren.

Detoxification of a heap (assuming cyanide leaching)

- determine whether to use hypochlorite, peroxide, or SO_2/air.
- determine the detox procedure.
- passivate glassware with nitric acid when assaying for cyanide species.
- keep good records of pH and Eh throughout detoxification cycle.
- preserve solution samples immediately upon taking them.
- assay detoxification solutions for metal values to help determine soluble losses.
- plot the concentration of cyanide species vs time throughout the detox cycle - free, WAD, total, thiocyanate, and cyanate.

Residues

- determine wet weight and the moisture content.

- observe whether the agglomerates are intact, smeared, or compressed; take photos.
- observe whether the residue is relatively dry or sloppy wet.
- assay the residues in about 5' vertical increments if a tall column was used or if the samples are taken from a test heap.
- keep the sample from each 5' vertical increment separate from the others during preparation and assaying.
- save a split of the wet residue for future washing tests or EP Toxicity tests, etc. Keep it moist.
- split out a sample of wet residue and wash it to determine soluble loss of values.
- perform a wet screen analysis and get assays of the sized fractions --- calculate a residue assay. Compare with similar screen analyses on fresh ore. Use same screen sizes as were used on fresh ore screen analyses.

<u>Test Heaps</u>

- are permits needed?
- agglomerate the ore, unless it is run-of-mine size.
- keep heavy equipment off the heap.
- if built with trucks, doze off upper 5' and then rip the surface before putting on spray system.
- if a stacker is used, keep it moving or make very small cones.
- spray side slopes.
- obtain backhoed samples from surface to bottom of heap when test is done - do this on a regular grid pattern.
- observe for ponding on surface of heap and correlate with observation of the final residue via backhoed trenches.
- take many head samples during crushing and/or agglomeration at regular intervals.
- give adequate weighting to the side slope ore when calculating extraction.
- calibrate the ponds so that good measurements of solution volume can be made.
- install good flowmeters and samplers and pumps.

<u>Calculations</u>

- extraction of values (account for all sample volumes sent to the lab, the wash volumes, the values adsorbed on carbon as compared to the preg - barren values)

- reagent consumption
- water balance
- detoxification reagent usage
- all the parts of the overall cycle time if reuseable leach pads are to be used

MET TESTING

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FORMERLY
COLORADO SCHOOL OF MINES
RESEARCH INSTITUTE

August 6, 1987

IPRC Project NP-872038

Mr. Rex Outzen
General Manager
Brohm Mining Corporation
P.O. Box 485
Deadwood SD 57732

Re: Metallurgical Studies on Gilt Edge Ore Samples

Dear Mr. Outzen:

International Process Research Corporation has completed preliminary metallurgical tests on three samples of Gilt Edge ore as proposed in our letter of May 21, 1987. Process evaluation included heavy-liquid separation, amalgamation for free gold, flotation, leaching of whole ore and of flotation concentrate, and Bond grindability tests.

SUMMARY

Each ore type contained the following quantity of gold and silver by direct fire assay.

	oz/ton	
	Gold	Silver
Sulfide Ore (S)	0.026	0.038
Mixed Sulfide and Oxidized Ore (M)	0.037	0.045
Oxidized Ore (O)	0.046	0.031

The potential for gravity separation was investigated by the use of heavy-liquid separation at 2.95 sp gr. The following data summarized the results.

Ore	Head Calculated Au	Sink Product Weight %	Au and Ag Distribution Sink %	Float %
Sulfide	0.036	4.8	43	57
Mixed	0.050	2.0	26	74
Oxidized	0.040	1.0	49	51

The above results were achieved at a -65 mesh grind. Oxides reponded most favorably of the three ores tested, but the results indicate that the ores will not respond well to gravity separation.

The presence of free gold was determined by amalgamation for each sample at a -65 mesh grind. The following results were obtained.

Ore	Head oz Au/ton	Gold Recovery in Amalgam %
Sulfide	0.026	19
Mixed	0.037	2
Oxide	0.046	22

The amalgamation results appear to parallel the heavy-liquid separation test results. Amalgamation supports the conclusion that these samples are not amenable to gravity separation for the recovery of gold.

Flotation studies were conducted on each ore type. Tests were conducted at grinds of -35, -65, and -100 mesh. A summary of results is shown below.

Ore	Grind	Head Calculated Au oz/ton	Concentrate Weight %	Concentrate Au oz/ton	Concentrate Au Recovery %	Tailing oz/ton
Sulfide	-35	0.031	11.6	0.19	71	0.010
	-65	0.058	10.5	0.48	87	0.008
	-100	0.029	10.1	0.21	72	0.009
Mixed	-35	0.055	6.9	0.60	75	0.015
	-65	0.047	8.4	0.40	72	0.014
	-100	0.046	8.7	0.41	76	0.012
Oxide	-35	0.047	2.7	0.70	40	0.029
	-65	0.050	4.2	0.59	50	0.026
	-100	0.048	3.2	0.68	45	0.027
Oxide	-65	0.047	6.6	0.38	53	0.024
	-65	0.045	5.7	0.44	55	0.021
Mixed	-65	0.041	7.2	0.36	72	0.012

Gold recovery from the sulfide and mixed ores was generally in the region of 71% to 76% with tailing assays of 0.008 to 0.01 oz/ton for sulfides and 0.012 to 0.015 oz/ton for mixed.

The oxide ore sample showed the poorest flotation response despite several procedure adjustments. Gold recovery was maximized at 55%. Tailing grades of 0.021 to 0.029 were typical.

Leach studies were conducted on whole ore and on sulfide flotation concentrate. The data from the whole ore leaching tests are shown below. The final extractions are at 72 hr.

Ore	Grind	Head Calculated Au oz/ton	Gold Extraction %	Leach Tailing Au oz/ton	Reagent Consumption NaCN lb/ton	Reagent Consumption $Ca(OH)_2$ lb/ton
Sulfide	-35	0.034	67	0.011	2.42	5.4
	-65	0.026	73	0.007	2.74	5.2
	-100	0.028	79	0.006	2.96	4.8
Mixed	-35	0.037	74	0.010	2.64	5.7
	-65	0.036	76	0.009	1.50	6.1
	-100	0.041	81	0.008	2.34	6.3
Oxide	-35	0.044	79	0.009	2.44	4.4
	-65	0.044	81	0.009	2.60	4.4
	-100	0.044	82	0.008	2.70	4.4

Gold extractions generally improved with increasing oxide ore content.

Leaching tests on flotation concentrate was conducted on material produced from the sulfide ore sample. Tests were conducted on roasted and unroasted concentrate samples. The results are shown below:

	Head Calculated Au oz/ton	Gold Extraction %	Tailing Au oz/ton
Roasted Concentrate	0.292	90	0.030
Nonroasted Concentrate	0.222	77	0.052

Roasting of the concentrate clearly enhanced the extraction. The combined metallurgical results on sulfide ore flotation and concentrate leaching are shown below.

	Weight %	Au Assay oz/ton	Au Distribution %
Head (calculated)	100.0	0.030	100.0
Flotation Tailing	90.0	0.009	26.9
Flotation Concentrate	10.0	0.22	73.1
Weight Loss (roasting)	2.5	0.0	0.0
Leach Feed	7.5	0.292	73.1
Leach Tailing	7.5	0.030	7.5
Pregnant Solution	--	--	65.6

Bond grindability tests were conducted on the ore samples. The results are shown below.

	Bond Work Index	
Ore	Rod Mill (at 14M)	Ball Mill (at 65M)
Sulfide	[1]	13.6
Mixed	[1]	12.7
Oxide	10.8	12.6

[1] Particle size distribution of sample was below the required -½ in. feed specification.

The grindability values are in a nominal range for hard rock ore. The oxide ore shows a slightly lower ball mill work index than the nonoxidized sample which is to be expected.

RECOMMENDATIONS

Because flotation of the Gilt Edge ore will be directed to the sulfide and possiblly mixed ores, future flotation shall be specific to the sulfide types. A review of the simple suitability should be made, and a new sample submitted if needed. Criteria for a suitable sample should include:

??

1. Precious metal content.
2. Geologic characterization.
3. Mineralogy.

Flotation was able to achieve tailing grades on the sulfide ore in the region of 0.008 to 0.010 oz Au/ton which resulted in gold recovery of 71% to 72% in an 0.03 oz/ton feed. If the same tailing grades can be maintained, 90% gold recovery should be achievable on 0.08 oz/ton ore. Additional flotation tests should be conducted to address the following:

1. Maximize Au and Ag recovery in a rougher/cleaner flotation system.

2. Simplify and minimize reagent consumption.

3. Minimize slime entrapment in the flotation concentrates.

4. Establish flotation rate curves from which to determine flotation cell requirements.

5. Confirm batch results conducting a lock-cycle flotation test for rougher and cleaner stages.

The gold extraction from the unroasted flotation concentrate was 77%. The extraction was very rapid and appeared to have reached almost final extraction in 2 hr. It is recommended to invest the influence of finer grinding of the concentrate with the objective of increasing gold recovery by better liberation. Emphasis should be directed to the nonroasting option because of process cost considerations.

Flotation concentrate thickening tests should be conducted to identify a suitable flocculant, the minimal amount required, and to establish preliminary design criteria for thickener sizing.

If filtration is being contemplated for solid/liquid separation of the leach solids, laboratory filtration tests should be included in the next phase of work. The tests will develop necessary design criteria for filter selection.

Figure 1 displays a conceptual process flowsheet for which the above recommendations apply.

PHASE II COST ESTIMATE

The cost for conducting the recommended process studies is estimated to be $11,200. This is a preliminary estimated based on anticipated process requirements. We look forward to your comments and input to structure future studies to your specific needs.

IPRC appreciates the opportunity to be of service to Brohm and look forward to further development on this interesting project.

Sincerely,

Robert J. Phillips

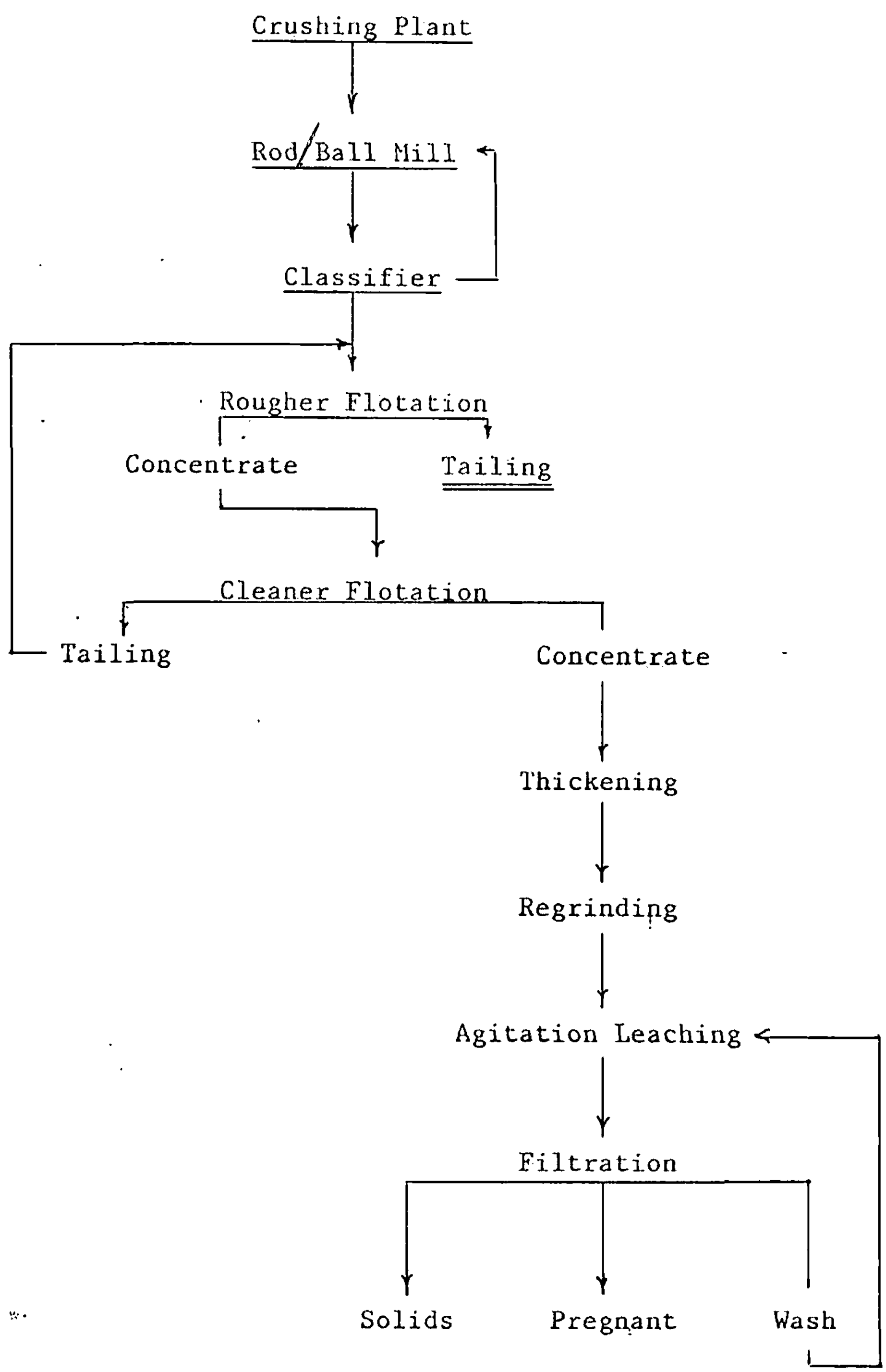
Robert J. Phillips
Chief Engineer

/psg

Enc.

FIGURE 1

Conceptual Process Flowsheet



DISCUSSION

SAMPLES

Three samples of ore were submitted for the project. The samples were labelled sulfide, mixed, and oxidized ore. One sample, oxide, was crushed to 100% passing ½ in. prior to subsequent blending and splitting. Exhibit 1 contains the sample descriptions.

ANALYSES

Analyses for the program were limited to gold and silver fire assays. Due to the presence of spotty gold values, 5-assay ton fire assays were run where sufficient sample was available. For samples that contained lesser amounts (<150 g), the total sample was assayed.

GRAVITY SEPARATION TESTS

Heavy liquid separations were conducted on each ore sample to predict probable effectiveness of gravity equipment for the recovery of gold and silver.

From each head sample, a representative 1,000 g was ground to -65 mesh and dried. A one-fourth split was used for a heavy-liquid separation at 2.95 sp gr. The resulting sink and float products were washed, dried, weighed, and assayed. The results of the tests are shown below in Table 1.

TABLE 1

Heavy-Liquid Separation Results

Sample S Product	Weight %	Chemical Analysis Au oz/ton	Chemical Analysis Ag oz/ton	Distribution % Au	Distribution % Ag
Head (calculated)	100.0	0.036	0.06	100.0	100.0
2.95 Sink	4.8	0.321	0.626	42.8	50.4
2.95 Float	95.2	0.022	0.031	57.2	49.6

Sample M Product	Weight %	Chemical Analysis Au oz/ton	Chemical Analysis Ag oz/ton	Distribution % Au	Distribution % Ag
Head (calculated)	100.0	0.051	0.078	100.0	100.0
2.95 Sink	2.0	0.655	0.838	25.9	21.6
2.95 Float	98.0	0.038	0.062	74.1	78.4

TABLE 1 -- continued

Sample O Product	Weight %	Chemical Analysis Au oz/ton	Chemical Analysis Ag oz/ton	Distribution % Au	Distribution % Ag
Head (calculated)	100.0	0.090	0.072	100.0	100.0
2.95 Sink	1.0	1.979	0.230	8.7	0.3
2.95 Float	99.0	0.021	0.071	91.3	99.7

The separation was most effective for the sulfide sample but still fell short of a satisfactory result. Based on these tests, the effectiveness of a gravity separation circuit seems negligible. Gravity separaton is not recommended on these samples.

FREE GOLD STUDIES

To supplement the gravity separation investigation, an amalgamation test was conducted on each sample to recover liberated gold. Amalgamation tests were conducted at -65 mesh. Parameters for the tests are listed below:

Solids, g:	1,000
NaOH:	6 pellets
Steel Balls:	5
Pulp Solids:	50
Mercury, g:	50
Run Time, hr:	24

Visible gold was detected in the amalgam residues after nitric acid digestion. The quantity of gold, however, accounted for only a minor part of the total as shown in Table 2.

TABLE 2

Amalgamation Results

Ore	Head (analyzed) Au oz/ton	Head (analyzed) Ag mg/1,000 g	Recovered Free Au mg	Gold Recovery %
S	0.03	1.03	0.177	16.6
M	0.037	1.27	0.026	2.1
O	0.046	1.57	0.365	23.2

The low gold recovery confirms the results of the heavy-liquid separation that free gold is not present in quantities suitable for gravity separation.

FLOTATION STUDIES

Flotation was conducted on each sample to establish the concentrate grade and gold recovery from the samples. Prior to testing, a laboratory rod mill was calibrated on each ore to establish correct grinding time for 100% -35 mesh, -65 mesh, and -100 mesh particle size distributions. A flotation procedure was established which was designed to recover free and oxidized sulfides. Pulp alkalinity was adjusted by sodium carbonate rather than lime to avoid the depressing effect of lime on gold and/or pyrite flotation. A standard reagent suite was used for the tests, and it is shown on the flotation data sheets in Exhibit 2.

Collectors were added to the rod mill, rougher flotation prior to sulfidization, and rougher flotation after sulfidization.

Three tests were conducted on each sample at -35, -65, and -100 mesh, respectively. Fire assays were conducted on the products. The results are shown in Exhibit 3.

Comments regarding the flotation results are as follows:

1. Flotation of the sulfide sample was more successful than that for the mixed and oxide samples in regard to gold recovery.

2. The additional particle liberation gained between 35 mesh and 100 mesh grinds resulted in very slight recovery improvement judging from the tailing grades.

3. The variation in calculated head grades was more influential on calculated recovery than the tailing assays.

4. For Sample M (mixed), sodium carbonate could not be added to the rod mill. The presence of Na_2CO_3 created a very viscous pulp. Sodium carbonate was added to the flotation cell after grinding. If clays are present that will react with certain reagents, this should be carefully taken into account in flowsheet design.

Additional flotation tests were conducted on mixed and oxide samples to improve gold recovery (Tests 10 through 12). The adjustment to the standard procedures are reflected in the test data sheets. Adjustment included:

1. Flotation on natural pH (lower).

2. Use of fatty acid to collect iron oxides that could partially contain gold values.

3. Evaluate desliming to enhance flotation selectivity.

4. Stage addition of sulfidization reagent.

The procedure modifications appeared to have no substantial impact on as evidenced by calculated gold recovery and by tailing grades. Comments regarding the tests are as follows:

1. Lower pH had no apparent benefit.

2. Fatty acid flotation of iron oxides improved gold recovery by approximately 9% (Test 11). A mineralogical exaimination of an oxide concentrate confirmed the presence of visible gold associated with the iron oxides. This is to be expected if the gold was originally associated with pyrite in the unoxidized ore.

3. Desliming of the oz Au/ton resulted in gold losses. The oxide ore slimes contained 0.040 (Test 11) and mixed ore slimes (Test 12) contained 0.022 oz Au/ton.

LEACHING STUDIES

Whole ore rolling bottle leaching tests were conducted to establish profiles for each sample. Three tests were conducted on each sample at -35, -65, and -100 mesh, respectively. Parameters for each test are shown below:

pH:	10.5+
NaCN, %:	0.1
Pulp Solids, %:	50
Total Leach Time, hr:	72
Liquid Samples, hr:	2, 4, 8, 24, 48, 72
Solids Sample, hr:	72

Figures 2 through 10 present the extraction profiles for whole ore leach tests. Data sheets for tests are contained in Exhibit 3.

Two leaching tests were conducted on sulfide ore flotation concentrate. The concentrate was pulverized to nominal -200 mesh and divided into two parts. One part was roasted in a muffle furnace for 4 min at 600°C. The second part was not roasted. The repulped solids were neutralized with lime prior to leaching.

Neutralization of the roasted concentrate required considerably more lime to achieve pH of 10.5 as compared to the lime needed for the unroasted sample. The lime consumption for each is shown below.

	Concentrate Weight g	Lime Weight g	lb Lime/ ton of solids
Roasted Concentrate	114.2	14.0	246.0
Unroasted Concentrate	152.0	1.5	19.7

Future tests on roasted material should include a water leach to remove the acid forming salts prior to neutralization.

Figures 11 and 12 present the extraction profiles from the roasted and nonroasted concentrates, respectively. For both tests, extraction was near completion after 2 hr. Gold extraction from the roasted concentrate was near 90% whereas extraction from the nonroasted sample was 77%. Future leaching studies on nonroasted concentrates should include the investigation of particle size and cyanide strength on gold recovery.

FIGURE 2

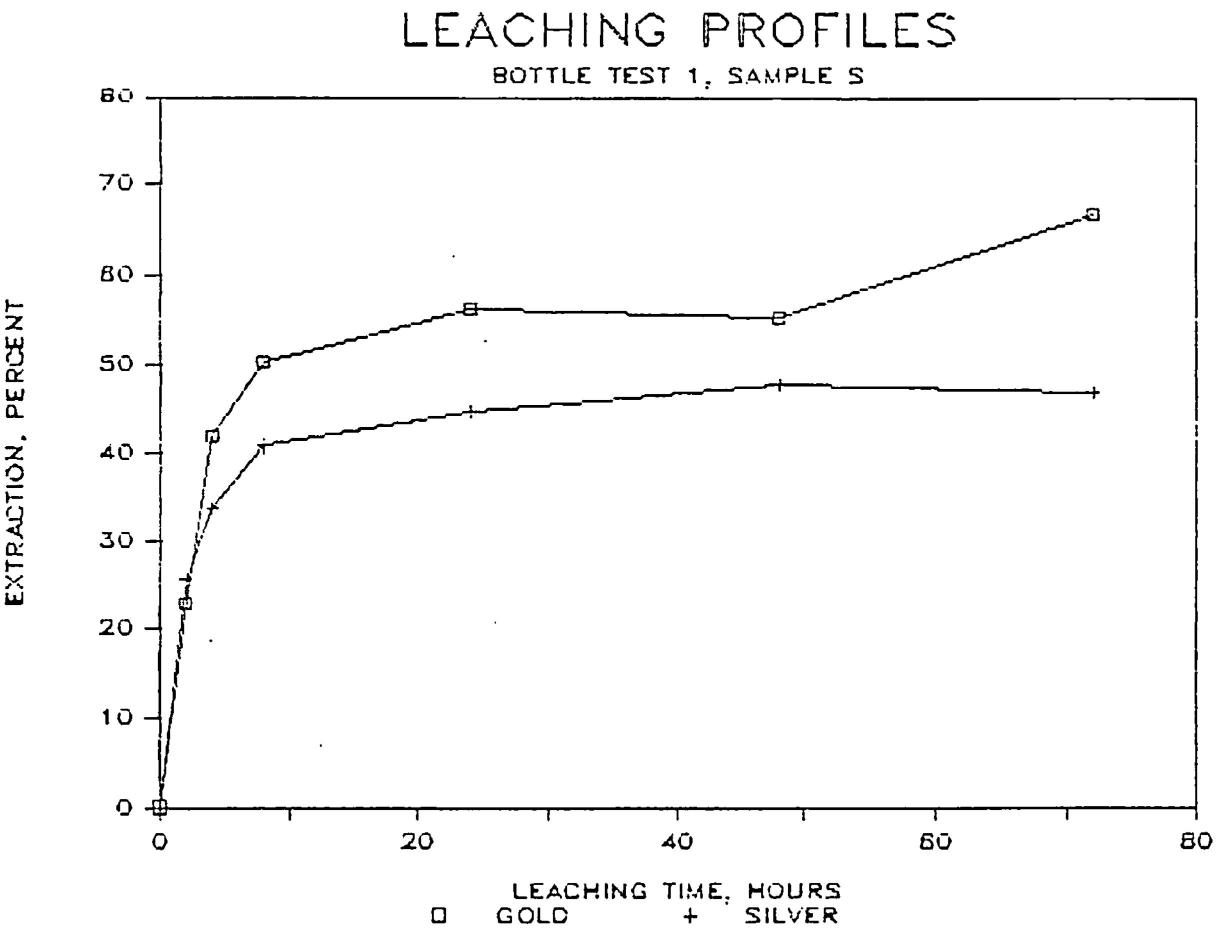

LEACHING PROFILES
BOTTLE TEST 1, SAMPLE S
EXTRACTION, PERCENT
0
10
20
30
40
50
60
70
80
0
20
40
60
80
LEACHING TIME, HOURS
GOLD
+ SILVER

FIGURE 3

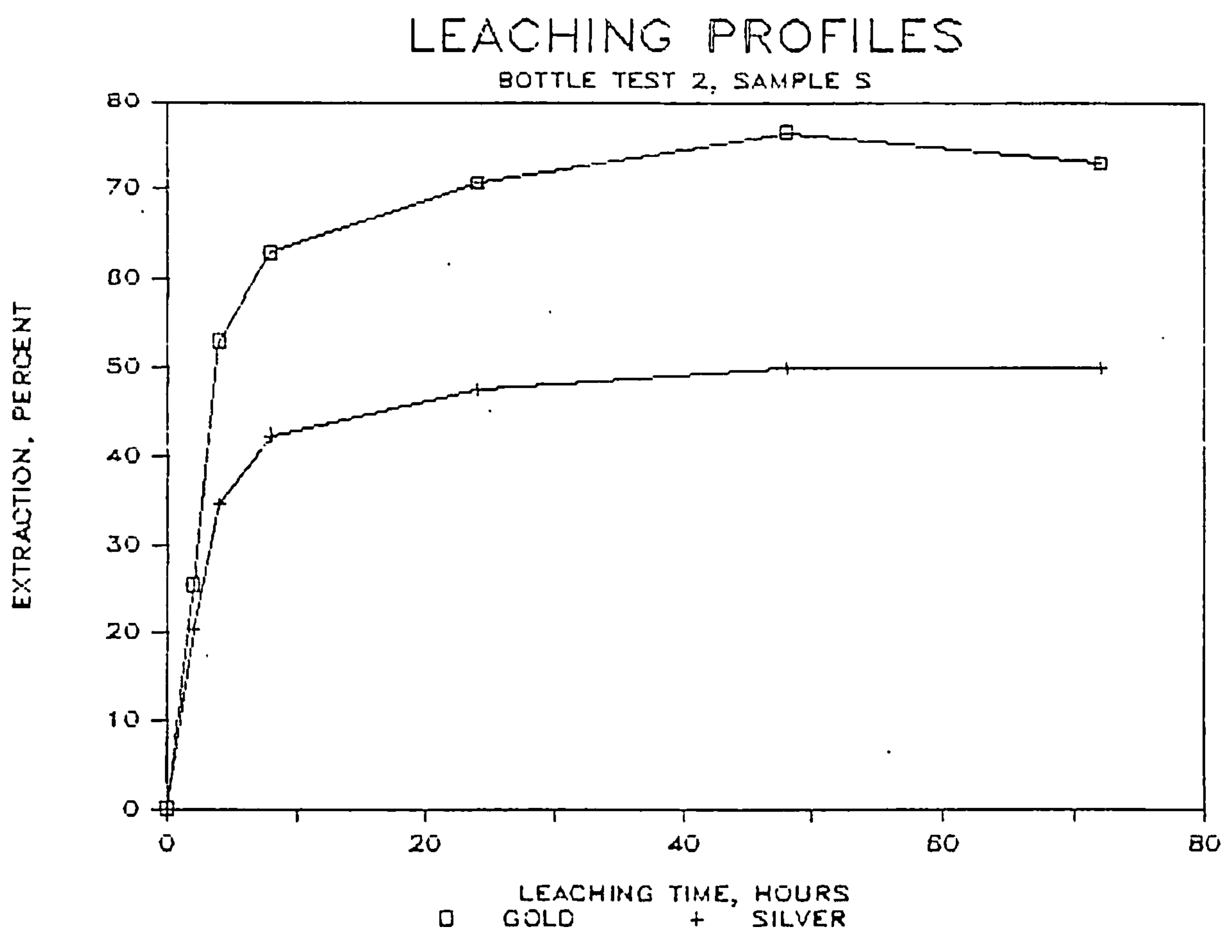

LEACHING PROFILES
BOTTLE TEST 2, SAMPLE S
EXTRACTION, PERCENT
0
10
20
30
40
50
60
70
80
0
20
40
60
80
LEACHING TIME, HOURS
□ GOLD + SILVER

FIGURE 4

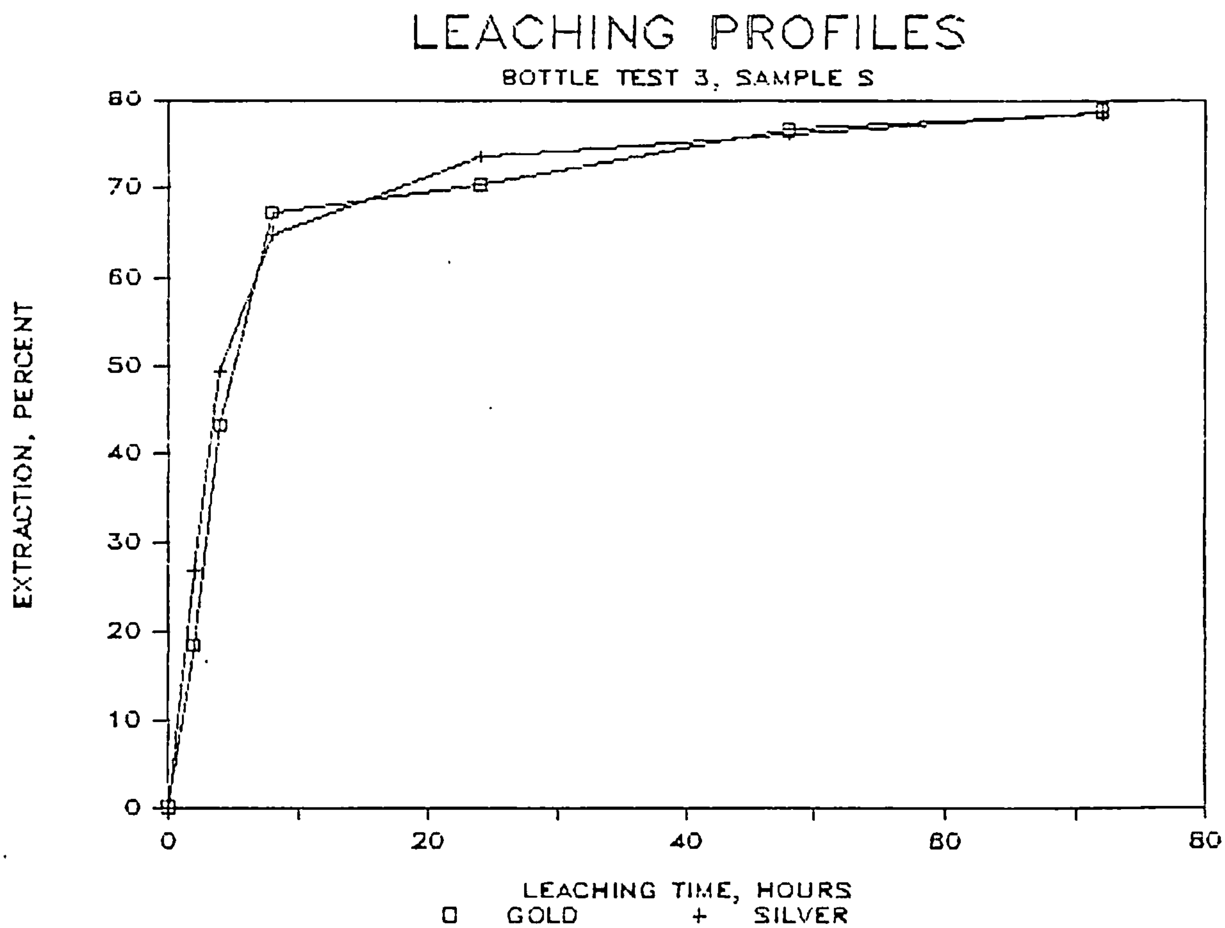

LEACHING PROFILES
BOTTLE TEST 3, SAMPLE S
EXTRACTION, PERCENT
0
10
20
30
40
50
60
70
80
0
20
40
60
80
LEACHING TIME, HOURS
GOLD
SILVER

FIGURE 5

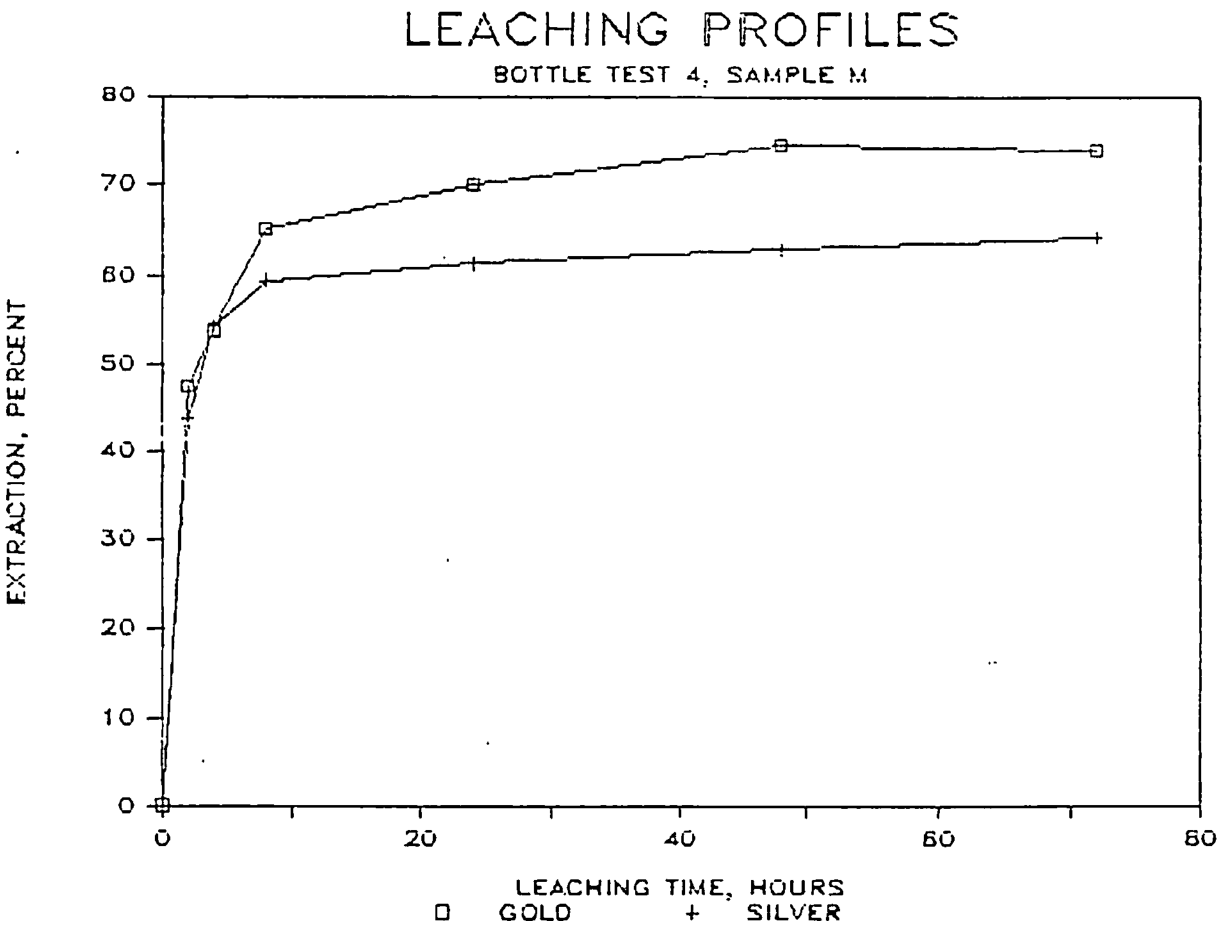

LEACHING PROFILES
BOTTLE TEST 4, SAMPLE M
EXTRACTION, PERCENT
80
70
60
50
40
30
20
10
0
0
20
40
60
80
LEACHING TIME, HOURS
GOLD
+ SILVER

FIGURE 6

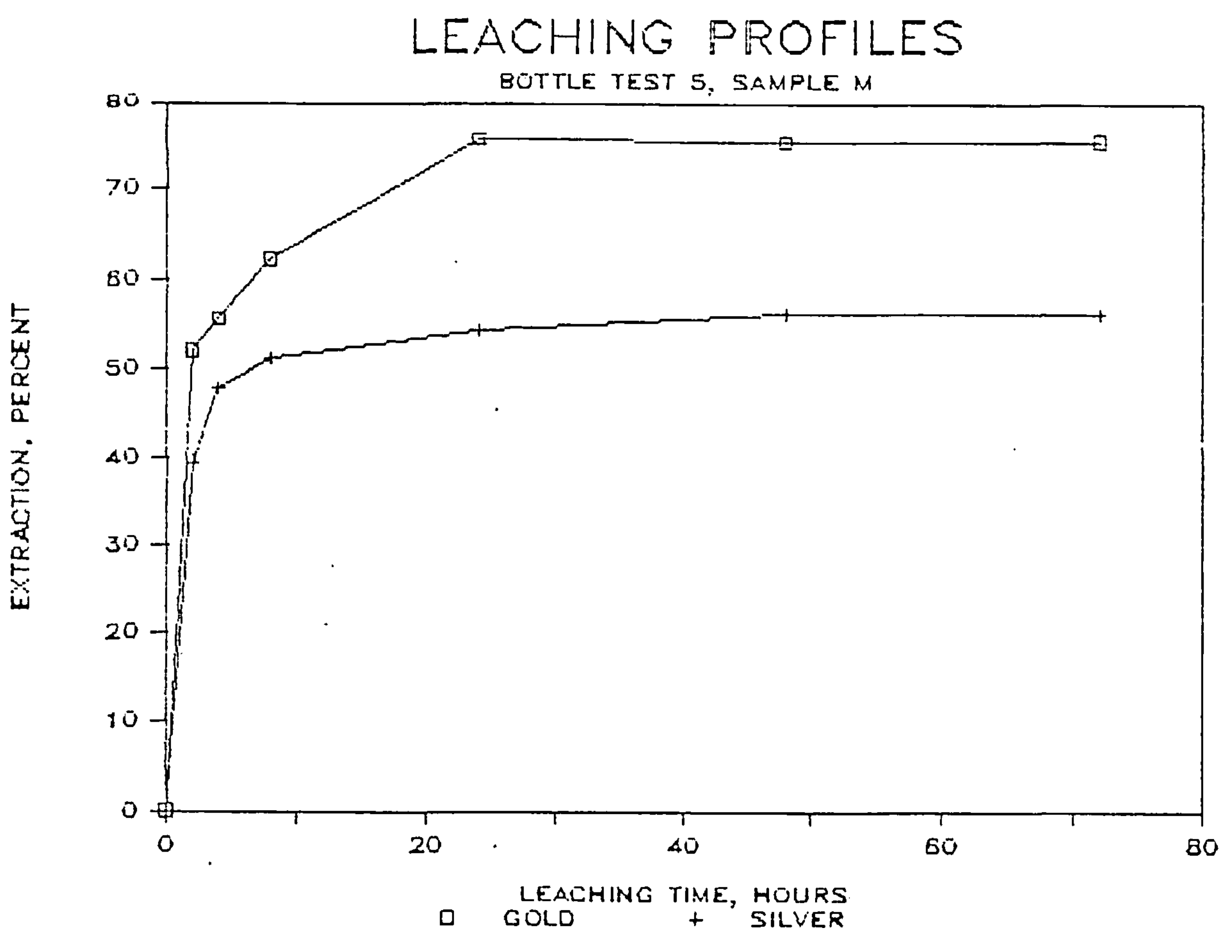

LEACHING PROFILES
BOTTLE TEST 5, SAMPLE M
EXTRACTION, PERCENT
0
10
20
30
40
50
60
70
80
0
20
40
60
80
LEACHING TIME, HOURS
GOLD
SILVER

FIGURE 7

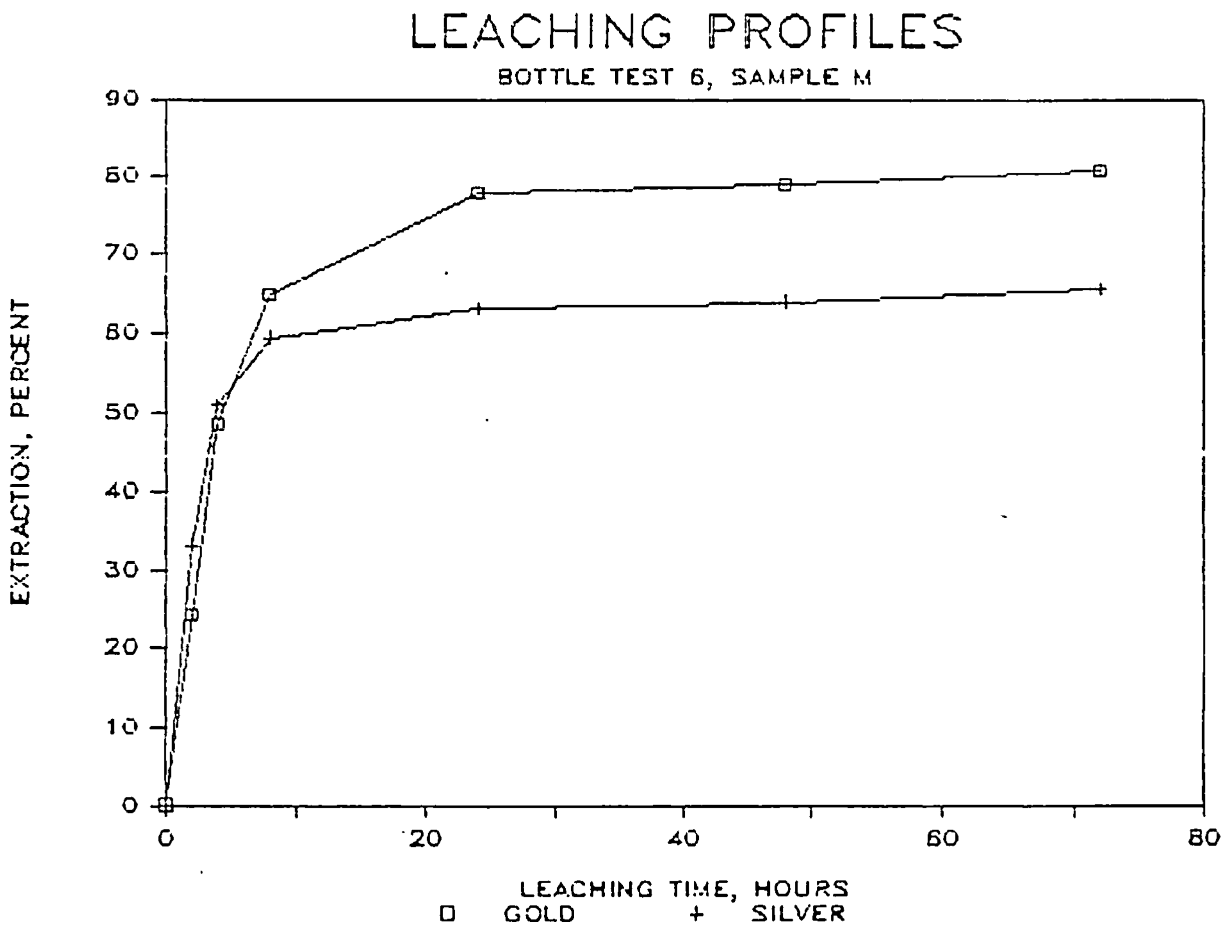

LEACHING PROFILES
BOTTLE TEST 6, SAMPLE M
EXTRACTION, PERCENT
90
80
70
60
50
40
30
20
10
0
0
20
40
60
80
LEACHING TIME, HOURS
GOLD
+ SILVER

FIGURE 8

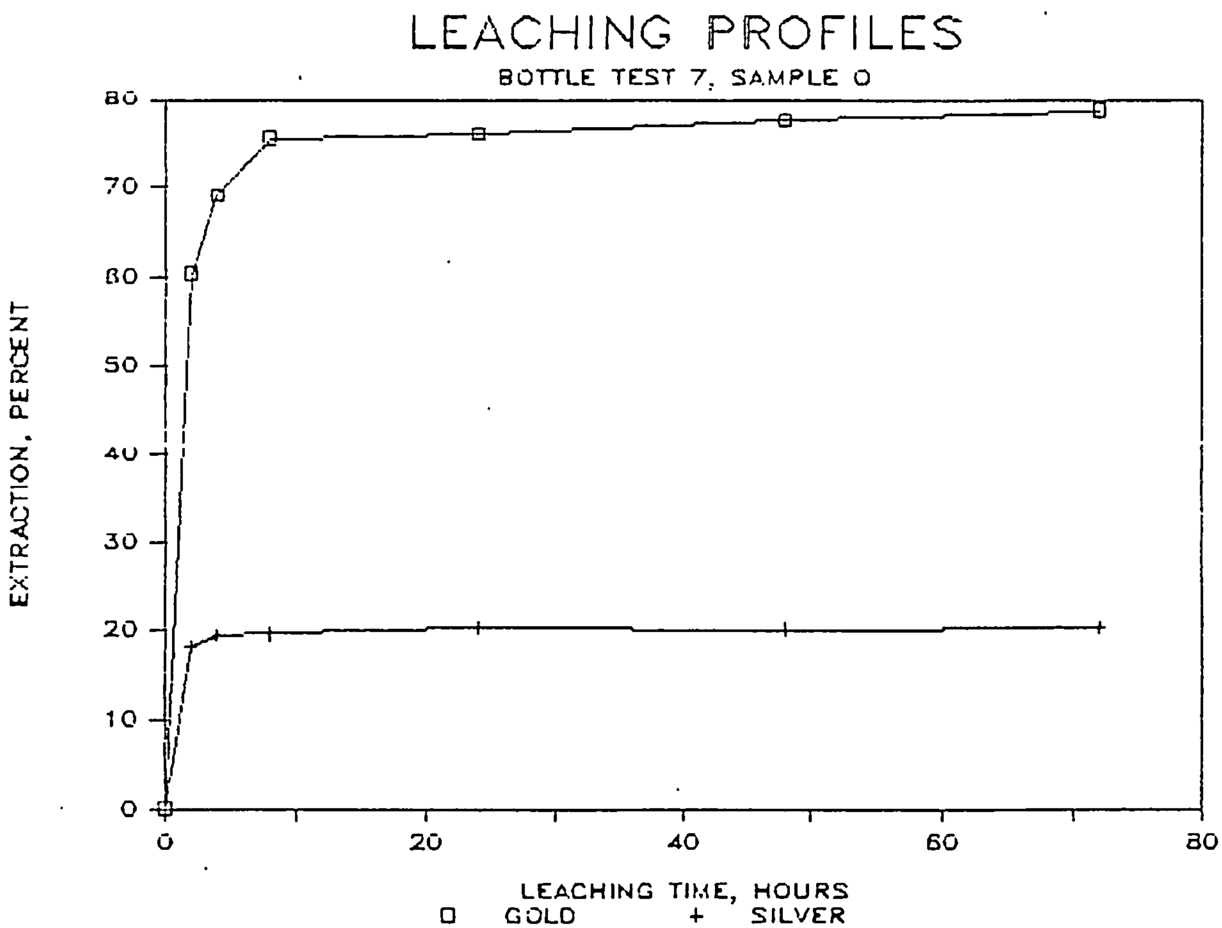

LEACHING PROFILES
BOTTLE TEST 7, SAMPLE O
EXTRACTION, PERCENT
0
10
20
30
40
50
60
70
80
0
20
40
60
80
LEACHING TIME, HOURS
GOLD
SILVER

FIGURE 9

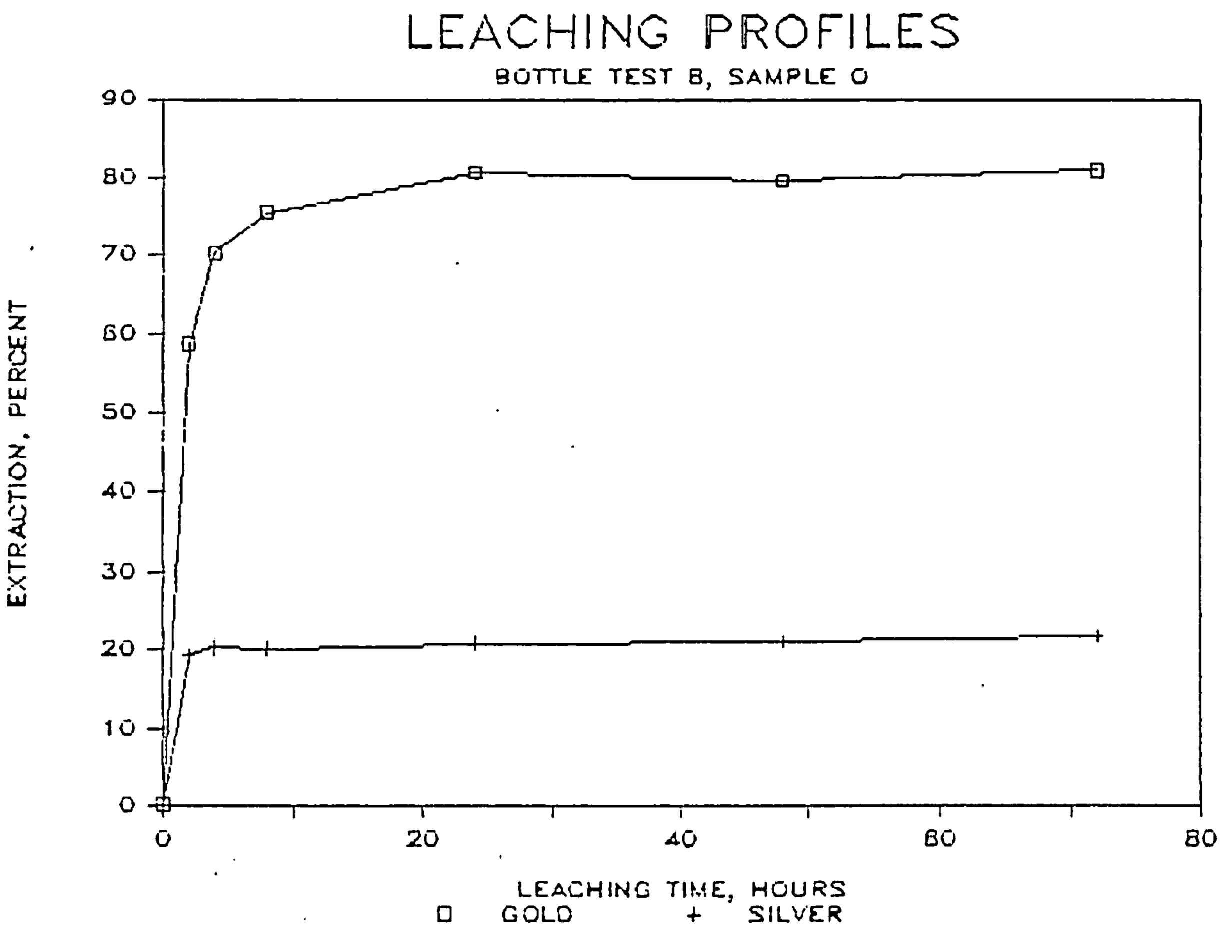

LEACHING PROFILES
BOTTLE TEST B, SAMPLE O
EXTRACTION, PERCENT
90
80
70
60
50
40
30
20
10
0
0
20
40
60
80
LEACHING TIME, HOURS
GOLD
SILVER

FIGURE 10

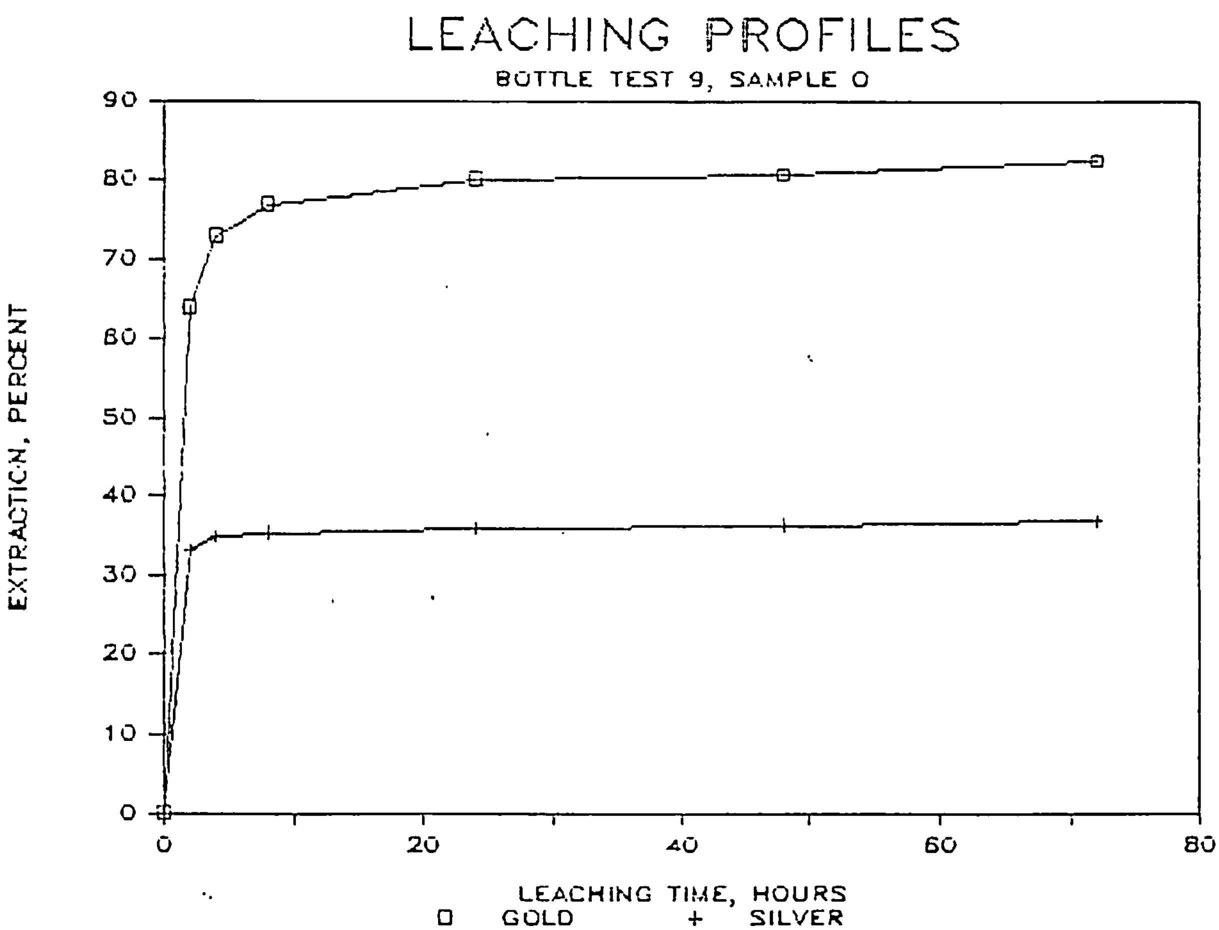

LEACHING PROFILES
BOTTLE TEST 9, SAMPLE O
EXTRACTION, PERCENT
0
10
20
30
40
50
60
70
80
90
0
20
40
60
80
LEACHING TIME, HOURS
GOLD
SILVER

FIGURE 11

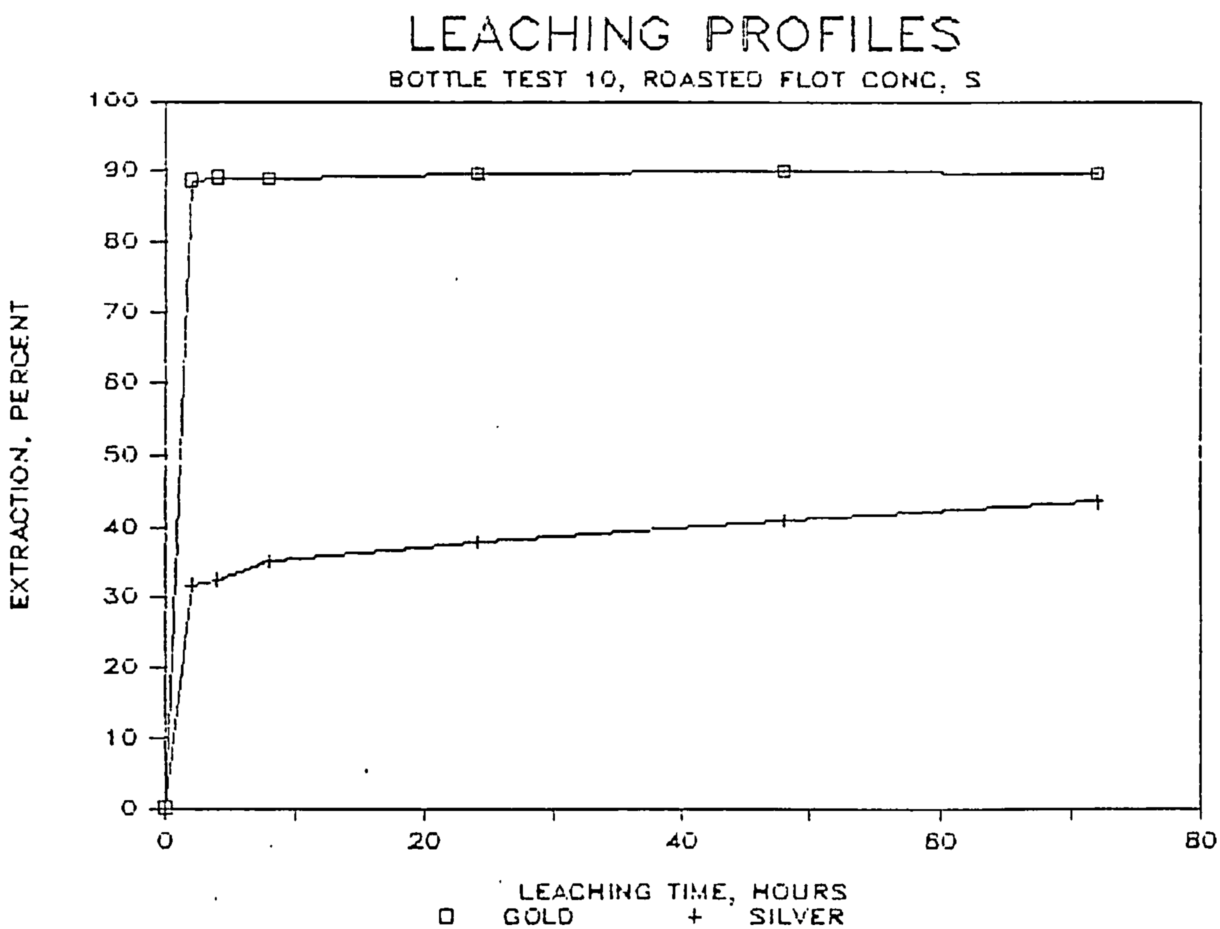
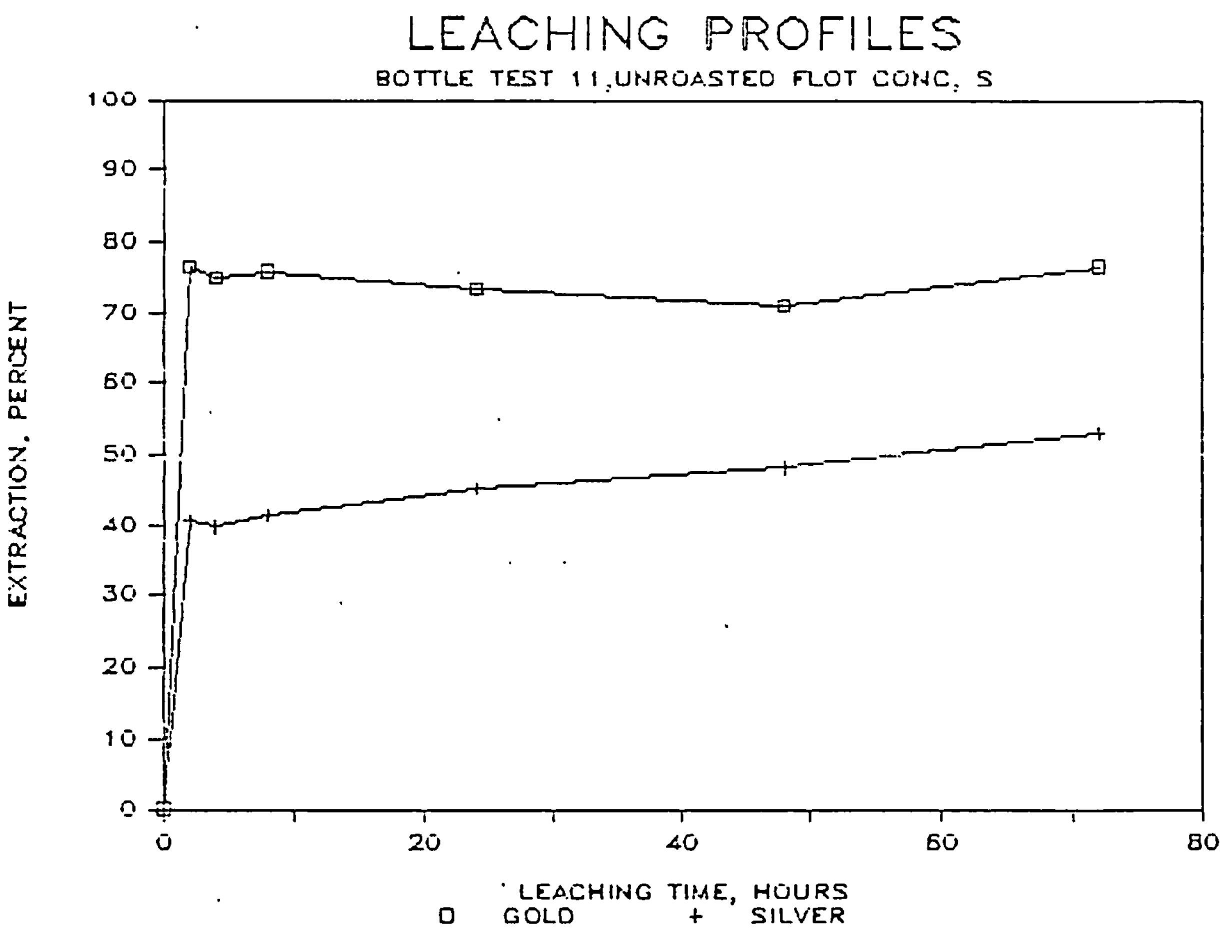

LEACHING PROFILES
BOTTLE TEST 10, ROASTED FLOT CONC, S
100
90
80
70
60
50
40
30
20
10
0
EXTRACTION, PERCENT
0
20
40
60
80
LEACHING TIME, HOURS
□ GOLD + SILVER

FIGURE 12


LEACHING PROFILES
BOTTLE TEST 11,UNROASTED FLOT CONC, S
EXTRACTION, PERCENT
100
90
80
70
60
50
40
30
20
10
0
0
20
40
60
80
LEACHING TIME, HOURS
□ GOLD + SILVER

GRINDABILITY TESTS

Rod and ball mill grindability tests were conducted in accordance with the Bond procedure. Of the three samples, only the oxide sample was of a size to permit rod mill testing. The other samples were too fine for a rod test. Ball mill grindability tests were conducted on each sample. The test mill for each grindability test is contained in Exhibit 4.

APPENDIX

EXHIBIT 1

SAMPLE DESCRIPTION AND PREPARATION

CSMRI Sample 1

Sponsor's Designation of Sample:	Sample S (sulfide ore).
Date Received at Institute:	June 17, 1987.
Sample Weight:	641 net.
Sample Container:	One steel drum.
Sample Description:	Visible pyrite, gray rock powder, approximately 75% -1/8 in., drill cuttings, dry.
Method of Preparation:	The sample was coned three times for blending. A ¼-in. split was removed and crushed to passing 10M. The -10M material was blended, and a head sample was split from it.

EXHIBIT 1

CSMRI Sample 2

Sponsor's Designation of Sample:	Sample M (mixed sulfide and oxide ore).
Date Received at Institute:	June 17, 1987.
Sample Weight:	515 lb net.
Sample Container:	One steel drum.
Sample Description:	Dried mud balls, approximately 75% -1 in.. gray, tan pink, white, sulfides visible.
Method of Preparation:	The sample was coned three times for blending. A ¼-in. split was removed and crushed to passing 10M. The -10M material was blended, and a head sample was split from it.

EXHIBIT 1

CSMRI Sample 3

Sponsor's Designation of Sample:	Sample O (oxide ore).
Date Received at Institute:	June 17, 1987.
Sample Weight:	Not recorded.
Sample Container:	One steel drum.
Sample Description:	-6M rock, rust red; aggregates of fine particles. Very slightly moist.
Method of Preparation:	The sample was screened at ½ in., and the oversize was crushed to ½ in. Samples were split from the bulk for grindability tests and metallurgical work.

EXHIBIT 2

FLOTATION TESTS

Flotation Test 1

Purpose: Determine the flotation response for gold and silver recovery.

Sample: Sample S, ground to nominal -35M.

Test Conditions:

	Time min	Solids %	pH Start	pH Finish	Reagents, lb/ton of feed AP-25	AP-404	AX-350	NaHS	$CuSO_4$	Frother MIBC	Na_2CO_3
Grinding (rod mill)	9.0	60	--	--	0.05	0.03	--	--	--	--	2.0
Conditioning	2.0	--	6.6	7.8	--	--	0.05	--	--	--	--
Flotation	4.0	--	--	--	--	--	--	--	--	--	--
Flotation	2+2	--	--	--	--	--	0.03+0.03	--	--	0.016	--
Conditioning	5.0	--	--	--	--	--	--	0.1	--	--	--
Conditioning	2.0	--	--	--	--	--	--	--	0.05	--	--
Flotation	4.0	--	--	7.8	--	0.035	0.035	--	--	0.008	--

Results:

Product	Weight %	Chemical Analysis Au oz/ton	Ag oz/ton	Distribution % Au	Ag
Head (calculated)	100.0	0.031	0.11	100.0	100.0
Rougher Concentrate	11.6	0.19	0.89	71.0	91.5
Rougher Tailing	88.4	0.010	0.012	29.0	9.5

EXHIBIT 2

Flotation Test 2

Purpose: Determine the flotation response for gold and silver recovery.

Sample: Sample S, ground to nominal -65M.

Test Conditions:

	Time min	Solids %	pH Start	pH Finish	Reagents, lb/ton of feed: AP-25	AP-404	AX-350	NaHS	$CuSO_4$	Frother MIBC	Na_2CO_3
Grinding (rod mill)	12.5	60	--	--	0.02	0.01	--	--	--	--	4.0
Conditioning	2.0	--	6.7	--	--	--	0.05	--	--	0.008	--
Flotation	4.0	--	--	--	--	--	--	--	--	--	--
Flotation	2+2	--	--	--	--	--	0.03+0.03	--	--	--	--
Conditioning	5.0	--	--	--	--	--	--	0.1	--	--	--
Conditioning	2.0	--	--	--	--	--	--	--	0.05	0.008	--
Flotation	4.0	--	--	7.8	--	0.030	0.030	--	--	--	--

Results:

Product	Weight %	Chemical Analysis: Au oz/ton	Chemical Analysis: Ag oz/ton	Distribution %: Au	Distribution %: Ag
Head (calculated)	100.0	0.058	0.276	100.0	100.0
Rougher Concentrate 1	8.9	0.544	2.811	83.3	90.7
Rougher Concentrate 2	1.6	0.148	0.825	4.2	4.8
Rougher Tailing	89.5	0.008	0.014	12.5	4.5

EXHIBIT 2

Flotation Test 3

Purpose: Determine the flotation response for gold and silver recovery.

Sample: Sample S, ground to nominal -100M.

Test Conditions:

	Time min	Solids %	pH Start	pH Finish	Reagents, lb/ton of feed AP-25	AP-404	AX-350	NaHS	$CuSO_4$	Frother MIBC	Na_2CO_3
Grinding (rod mill)	15.5	60	--	--	0.02	0.01	--	--	--	--	4.0
Conditioning	2.0	--	7.0	--	--	--	0.05	--	--	0.008	--
Flotation	4.0	--	--	--	--	--	--	--	--	--	--
Flotation	2+2	--	--	--	--	--	0.03+0.03	--	--	--	--
Conditioning	5.0	--	7.4	--	--	--	--	0.1	--	--	--
Conditioning	2.0	--	--	--	--	--	--	--	0.05	0.008	--
Flotation	4.0	--	--	--	--	0.030	0.030	--	--	--	--

Results:

Product	Weight %	Chemical Analysis Au oz/ton	Chemical Analysis Ag oz/ton	Distribution % Au	Distribution % Ag
Head (calculated)	100.0	0.029	0.089	100.0	100.0
Rougher Concentrate	10.1	0.210	0.795	72.4	89.9
Rougher Tailing	89.9	0.009	0.010	27.6	10.1

EXHIBIT 2

Flotation Test 4

Purpose: Determine the flotation response for gold and silver recovery.

Sample: Sample M, ground to nominal -35M.

Test Conditions:

	Time min	Solids %	pH Start	pH Finish	Reagents, lb/ton of feed AP-25	AP-404	AX-350	NaHS	$CuSO_4$	Frother MIBC	Na_2CO_3
Grinding (rod mill)	5.5	60	--	--	0.02	0.01	--	--	--	--	--
Conditioning	2.0	--	5.3	8.3	--	--	0.05	--	--	--	2.0
Flotation	4.0	--	--	--	--	--	--	--	--	0.016	--
Flotation	2+2	--	--	--	--	--	0.03+0.03	--	--	0.008	--
Conditioning	5.0	--	--	--	--	--	--	0.1	--	--	--
Conditioning	2.0	--	--	--	--	--	--	--	0.05	--	--
Flotation	4.0	--	--	7.6	--	0.030	0.030	--	--	0.008	--

Results:

Product	Weight %	Chemical Analysis Au oz/ton	Ag oz/ton	Distribution % Au	Ag
Head (calculated)	100.0	0.554	0.160	100.0	100.0
Rougher Concentrate	6.9	0.600	1.795	74.7	77.3
Rougher Tailing	93.1	0.015	0.039	25.3	22.7

EXHIBIT 2

Flotation Test 5

Purpose: Determine the flotation response for gold and silver recovery.

Sample: Sample M, ground to nominal -65M.

Test Conditions:

	Time min	Solids %	pH Start	pH Finish	Reagents, lb/ton of feed AP-25	AP-404	AX-350	NaHS	$CuSO_4$	Frother MIBC	Na_2CO_3
Grinding (rod mill)	10.0	60	--	--	0.02	0.01	--	--	--	--	--
Conditioning	2.0	--	6.6	7.9	--	--	0.05	--	--	--	0.5
Flotation	4.0	--	--	--	--	--	--	--	--	0.016	--
Flotation	2+2	--	--	--	--	--	0.03+0.03	--	--	0.008	--
Conditioning	5.0	--	--	--	--	--	--	0.1	--	--	--
Conditioning	2.0	--	--	--	--	--	--	--	0.05	0.008	--
Flotation	4.0	--	--	7.9	--	0.030	0.030	--	--	--	--

Results:

Product	Weight %	Chemical Analysis Au oz/ton	Ag oz/ton	Distribution % Au	Ag
Head (calculated)	100.0	0.047	0.134	100.0	100.0
Rougher Concentrate 1	6.4	0.492	1.338	67.5	63.7
Rougher Concentrate 2	2.0	0.121	0.373	5.1	5.6
Rougher Tailing	91.6	0.014	0.045	27.4	30.7

EXHIBIT 2

Flotation Test 6

Purpose: Determine the flotation response for gold and silver recovery.

Sample: Sample M, ground to nominal -100M.

Test Conditions:

	Time min	Solids %	pH Start	pH Finish	Reagents, lb/ton of feed AP-25	AP-404	AX-350	NaHS	$CuSO_4$	Frother MIBC	Na_2CO_3
Grinding (rod mill)	15.0	60	--	--	0.02	0.01	--	--	--	--	--
Conditioning	2.0	--	6.8	7.8	--	--	0.05	--	--	0.024	0.5
Flotation	4.0	--	--	--	--	--	--	--	--	--	--
Flotation	2+2	--	--	--	--	--	0.03+0.03	--	--	--	--
Conditioning	5.0	--	--	--	--	--	--	0.1	--	--	--
Conditioning	2.0	--	--	--	--	--	--	--	0.05	--	--
Flotation	4.0	--	--	7.4	--	0.03	0.03	--	--	0.008	--

Results:

Product	Weight %	Chemical Analysis Au oz/ton	Ag oz/ton	Distribution % Au	Ag
Head (calculated)	100.0	0.046	0.098	100.0	100.0
Rougher Concentrate	8.7	0.407	0.823	76.3	73.5
Rougher Tailing	91.3	0.012	0.028	23.7	26.5

EXHIBIT 2

Flotation Test 7

Purpose: Determine the flotation response for gold and silver recovery.

Sample: Sample O, ground to nominal -35M.

Test Conditions:

	Time min	Solids %	pH Start	pH Finish	Reagents, lb/ton of feed AP-25	AP-404	AX-350	NaHS	$CuSO_4$	Frother MIBC	Na_2CO_3
Grinding (rod mill)	8.5	60	8.9	--	0.02	0.01	--	--	--	--	--
Conditioning	2.0	--	--	--	--	--	0.05	--	--	--	1.0
Flotation	4.0	--	--	--	--	--	--	--	--	0.032	--
Flotation	2+2	--	--	--	--	--	0.03+0.03	--	--	0.016	--
Conditioning	5.0	--	--	--	--	--	--	0.1	--	--	--
Conditioning	2.0	--	--	--	--	--	--	--	0.05	0.008	--
Flotation	4.0	--	8.7	--	--	0.030	0.030	--	--	--	--

Results:

Product	Weight %	Chemical Analysis Au oz/ton	Chemical Analysis Ag oz/ton	Distribution % Au	Distribution % Ag
Head (calculated)	100.0	0.047	0.027	100.0	100.0
Rougher Concentrate	2.7	0.700	0.265	40.1	27.0
Rougher Tailing	97.3	0.029	0.020	59.9	73.0

EXHIBIT 2

Flotation Test 8

Purpose: Determine the flotation response for gold and silver recovery.

Sample: Sample O, ground to nominal -65M.

Test Conditions:

	Time min	Solids %	pH Start	pH Finish	Reagents, lb/ton of feed AP-25	AP-404	AX-350	NaHS	$CuSO_4$	Frother MIBC	Na_2CO_3
Grinding (rod mill)	15.5	60	--	--	0.02	0.01	--	--	--	--	1.0
Conditioning	2.0	--	--	8.7	--	--	0.05	--	--	0.016	--
Flotation	4.0	--	--	--	--	--	--	--	--	0.016	--
Flotation	2+2	--	--	--	--	--	0.03+0.03	--	--	0.016	--
Conditioning	5.0	--	--	--	--	--	--	0.1	--	--	--
Conditioning	2.0	--	--	--	--	--	--	--	0.05	--	--
Flotation	4.0	--	8.7	--	--	0.030	0.030	--	--	--	--

Results:

Product	Weight %	Chemical Analysis Au oz/ton	Chemical Analysis Ag oz/ton	Distribution % Au	Distribution % Ag
Head (calculated)	100.0	0.050	0.048	100.0	100.0
Rougher Concentrate 1	3.0	0.730	0.313	44.0	19.6
Rougher Concentrate 2	1.2	0.253	0.264	6.0	6.6
Rougher Tailing	95.8	0.026	0.037	50.0	73.8

EXHIBIT 2

Flotation Test 9

Purpose: Determine the flotation response for gold and silver recovery.

Sample: Sample O, ground to nominal -100M.

Test Conditions:

	Time min	Solids %	pH Start	pH Finish	Reagents, lb/ton of feed AP-25	AP-404	AX-350	NaHS	$CuSO_4$	Frother MIBC	Na_2CO_3
Grinding (rod mill)	20.0	60	--	--	0.02	0.01	--	--	--	--	1.0
Conditioning	2.0	--	8.7	--	--	--	0.05	--	--	0.016	--
Flotation	4.0	--	--	--	--	--	--	--	--	0.016	--
Flotation	2+2	--	--	--	--	--	0.03+0.03	--	--	0.008	--
Conditioning	5.0	--	--	--	--	--	--	0.1	--	--	--
Conditioning	2.0	--	--	--	--	--	--	--	0.05	--	--
Flotation	4.0	--	--	8.8	--	0.030	0.030	--	--	--	--

Results:

Product	Weight %	Chemical Analysis Au oz/ton	Ag oz/ton	Distribution % Au	Ag
Head (calculated)	100.0	0.048	0.040	100.0	100.0
Rougher Concentrate	3.2	0.676	0.309	45.3	24.8
Rougher Tailing	96.8	0.027	0.031	54.7	75.2

EXHIBIT 2

Flotation Test 10

Purpose: Determine the flotation response for gold and silver recovery.

Sample: Sample O, ground to nominal -65M.

Test Conditions:

	Time min	Solids %	pH Start	pH Finish	Reagents, lb/ton of feed AP-25	AP-404	AX-350	Na_2S	$CuSO_4$	Frother MIBC	Na_2CO_3
Grinding (rod mill)	15.0	60	7.2	--	0.02	0.01	--	--	--	--	--
Conditioning	2.0	--	--	--	--	--	0.05	--	--	0.016	--
Flotation	4.0	--	--	7.4	--	--	--	--	--	--	--
Flotation	2+2	--	7.5	--	--	--	0.03+0.03	--	--	0.016	--
Conditioning	10.0	--	--	--	--	--	--	[1]	--	--	--
Flotation	4.0	--	--	8.0	--	0.06	--	--	--	0.016	--

[1] Sulfidization: Used sufficient Na_2S to hold +350 mv for 10 min.

Results:

Product	Weight %	Chemical Analysis Au oz/ton	Chemical Analysis Ag oz/ton	Distribution % Au	Distribution % Ag
Head (calculated)	100.0	0.047	0.042	100.0	100.0
Rougher Concentrate 1	4.4	0.515	0.125	47.9	13.1
Rougher Concentrate 2	2.2	0.113	0.173	4.9	9.0
Rougher Tailing	93.4	0.024	0.035	47.2	77.9

EXHIBIT 2

Flotation Test 11

Purpose: Determine the flotation response for gold and silver recovery.

Sample: Sample O, ground to nominal -65M.

Test Conditions:

	Time min	Solids %	pH Start	pH Finish	AP-25	AP-404	AX-350	Na_2S	$CuSO_4$	Frother MIBC	Fatty Acid
					Reagents, lb/ton of feed						
Grinding (rod mill)	15.0	60	--	--	0.02	0.01	--	--	--	--	--
Conditioning	2.0	--	7.0	--	--	--	0.05	--	--	0.032	--
Flotation	4.0	--	--	7.6	--	--	--	--	--	--	--
Flotation	2+2	--	--	--	--	--	0.03+0.03	--	--	0.024	--
Conditioning	5.0	--	--	--	--	--	--	1	--	--	--
Conditioning	2.0	--	--	--	--	--	--	--	0.05	--	--
Flotation	4.0	--	--	--	--	0.05	--	--	--	--	--
FA Conditioning	5.0	70	--	--	--	--	--	--	--	0.008	0.08
FA Flotation	2.0	--	--	--	--	0.05	--	--	--	0.008	--

1 Used sufficient Na_2S to hold +325 mv for 10 min.

Results:

Product	Weight %	Chemical Analysis Au oz/ton	Chemical Analysis Ag oz/ton	Distribution % Au	Distribution % Ag
Head (calculated)	100.0	0.045	0.058	100.0	100.0
Rougher Concentrate 1	4.7	0.439	0.160	46.0	12.9
Rougher Concentrate 2	1.0	0.423	0.092	9.4	1.6
Rougher Tailing	65.7	0.013	0.052	19.1	58.6
Decanted Slime After First Flotation	28.6	0.040	0.055	25.5	26.9

EXHIBIT 2

Flotation Test 12

Purpose: Determine the flotation response for gold and silver recovery.

Sample: Sample M, ground to nominal -65M.

Test Conditions:

	Time min	Solids %	pH Start	pH Finish	Reagents, lb/ton of feed AP-25	AP-404	AX-350	Na_2S	$CuSO_4$	Frother MIBC	$NaSiO_2$	Fatty Acid
Grinding (rod mill)	10.0	60	--	--	0.02	0.01	--	--	--	--	0.5	--
Conditioning	2.0	--	5.8	--	--	--	0.05	--	--	0.032	--	--
Flotation	4.0	--	--	--	--	--	--	--	--	--	--	--
Flotation	2+2	--	5.6	--	0.03+0.03	--	--	--	--	0.008	--	--
Condition	5.0	--	--	--	--	[1]	--	--	--	--	--	--
Flotation	4.0	--	--	7.6	0.030	--	--	--	--	0.064	--	--
FA Conditioning	5.0	--	--	--	--	--	--	--	--	0.016	--	0.08
FA Flotation	2.0	--	7.3	7.4	--	--	--	--	--	--	--	--

[1] Used sufficient Na_2S to hold +350 mv for 10 min.

Results:

Product	Weight %	Chemical Analysis Au oz/ton	Ag oz/ton	Distribution % Au	Ag
Head (calculated)	100.0	0.041	0.129	100.0	100.0
Concentrate 1	5.6	0.361	0.724	50.0	31.4
Concentrate 2	1.6	0.440	1.183	17.4	14.7
Concentrate 3	1.0	0.189	0.369	4.7	2.9
Combined Slimes	32.3	0.022	0.145	17.6	36.3
Final Tailing	59.5	0.007	0.032	10.3	14.7

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EXHIBIT 2

Flotation Test 12 Flow Sheet

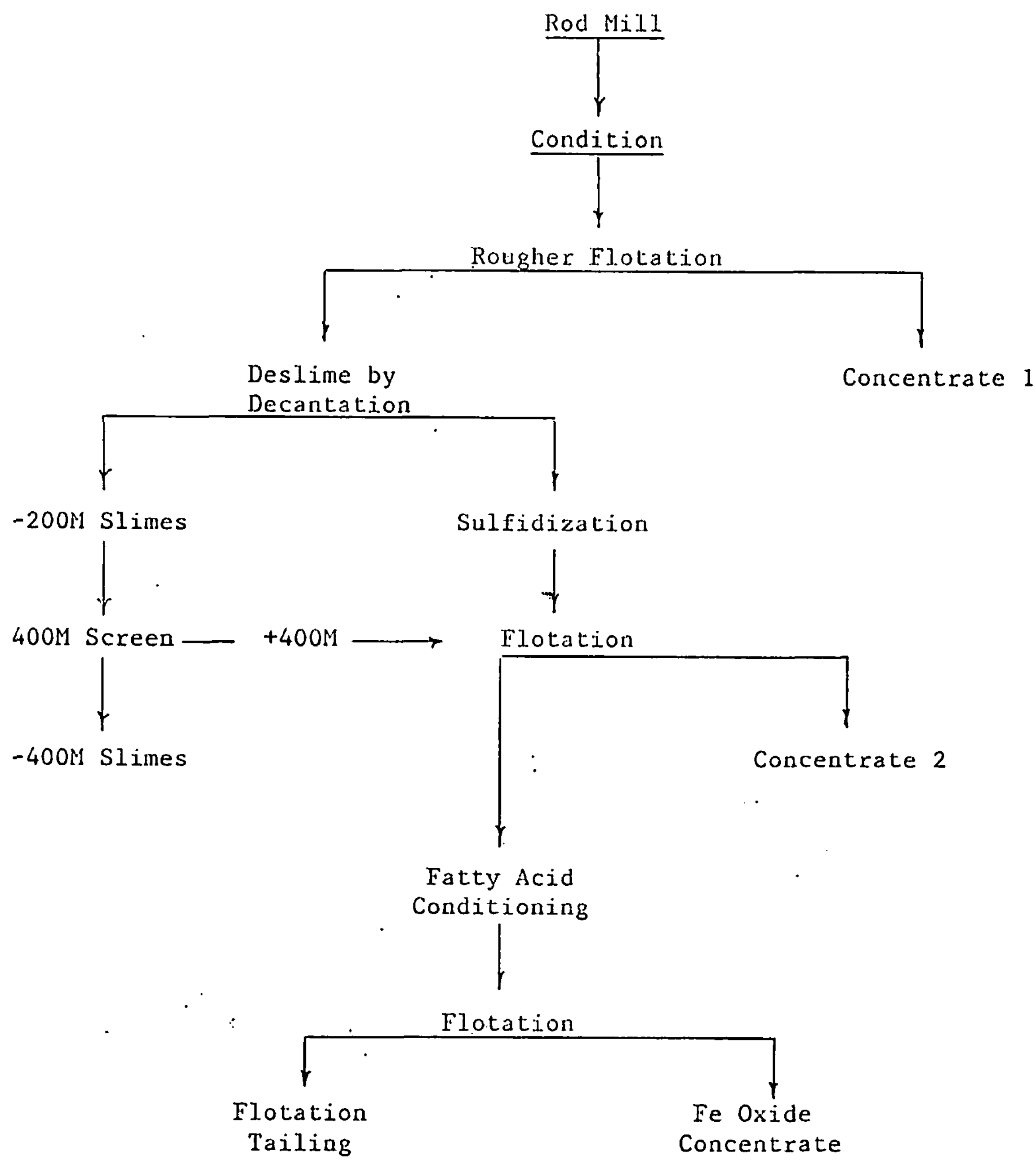


EXHIBIT 3

LEACHING TESTS

Cyanide Leaching Test 1
Sample: SAMPLE S, -35M GRIND

Results:
Reagent Consumption
NaCN, lb/ton 2.42
Ca(OH)2, lb/ton 5.4

Product	Weight g	Analysis Au oz/ton	Au ppm	Au mg(1)	Ag oz/ton	Ag ppm	Ag mg(1)	Distribution Au %	Ag %
Feed (analyzed)			--	--		--	--		
Feed (calculated)	984.2	0.034	--	1.15	0.11	--	3.56	100.0	100.0
Final Preg Soln	1016.6	--	0.72	0.77	--	1.55	1.67	66.6	46.9
Leached Residue	984.2	0.011	--	0.38	0.06	--	1.89	33.4	53.1
Preg Soln, hr									
0	1015.8	--	0.00	0.00	--	0.00	0.00	0.0	0.0
2	1016.7	--	0.26	0.26	--	0.90	0.92	22.9	25.7
4	1016.2	--	0.47	0.48	--	1.17	1.20	41.8	33.8
8	1016.4	--	0.56	0.58	--	1.40	1.45	50.3	40.8
24	1016.7	--	0.62	0.65	--	1.52	1.59	56.2	44.8
48	1017.9	--	0.60	0.64	--	1.60	1.70	55.3	47.7
72	1016.6	--	0.72	0.77	--	1.55	1.67	66.6	46.9

(1) Cumulative mg accounts for mg removed in sampling.

Cyanide Leaching Test 2
Sample: SAMPLE S, -65M GRIND

Results:
Reagent Consumption

NaCN, lb/ton	2.74
Ca(OH)2, lb/ton	5.2

Product	Weight g	Analysis Au oz/ton	Au ppm	Au mg(1)	Ag oz/ton	Ag ppm	Ag mg(1)	Distribution Au %	Ag %
Feed (analyzed)			--	--		--	--		
Feed (calculated)	985.4	0.026	--	0.87	0.10	--	3.38	100.0	100.0
Final Preg Soln	1013.5	--	0.59	0.64	--	1.57	1.69	72.9	50.0
Leached Residue	985.4	0.007	--	0.24	0.05	--	1.69	27.1	50.0
Preg Soln, hr									
0	1014.5	--	0.00	0.00	--	0.00	0.00	0.0	0.0
2	1014.5	--	0.22	0.22	--	0.68	0.69	25.6	20.4
4	1016.4	--	0.45	0.46	--	1.14	1.17	52.9	34.6
8	1014.0	--	0.53	0.55	--	1.38	1.43	62.9	42.3
24	993.1	--	0.60	0.62	--	1.56	1.61	70.8	47.5
48	1012.5	--	0.63	0.67	--	1.59	1.69	76.5	49.9
72	1013.5	--	0.59	0.64	--	1.57	1.69	72.9	50.0

(1) Cumulative mg accounts for mg removed in sampling.

Cyanide Leaching Test 3
Sample: SAMPLE S, -100M GRIND

Results:

Reagent Consumption

NaCN, lb/ton	2.96
$Ca(OH)_2$, lb/ton	4.8

Product	Weight	Analysis						Distribution	
		Au			Ag				
	g	oz/ton	ppm	mg(1)	oz/ton	ppm	mg(1)	Au %	Ag %
Feed (analyzed)			--	--		--	--		
Feed (calculated)	986.1	0.028	--	0.94	0.07	--	2.35	100.0	100.0
Final Preg Soln	1011.0	--	0.69	0.74	--	1.72	1.84	78.7	78.4
Leached Residue	986.1	0.006	--	0.20	0.02	--	0.51	21.3	21.6
Preg Soln, hr									
0	1013.9	--	0.00	0.00	--	0.00	0.00	0.0	0.0
2	1017.1	--	0.17	0.17	--	0.62	0.63	18.5	26.8
4	997.3	--	0.40	0.40	--	1.15	1.16	43.0	49.4
8	1017.0	--	0.61	0.63	--	1.47	1.53	67.4	65.0
24	1015.4	--	0.63	0.66	--	1.65	1.73	70.4	73.6
48	1012.4	--	0.68	0.72	--	1.69	1.79	76.6	76.1
72	1011.0	--	0.69	0.74	--	1.72	1.84	78.7	78.4

(1) Cumulative mg accounts for mg removed in sampling.

Cyanide Leaching Test 4
Sample: SAMPLE M, -35M GRIND

Results:
Reagent Consumption
NaCN, lb/ton 2.64
$Ca(OH)_2$, lb/ton 5.7

Product	Weight g	Analysis Au oz/ton	Au ppm	Au mg(1)	Ag oz/ton	Ag ppm	Ag mg(1)	Distribution Au %	Distribution Ag %
Feed (analyzed)			--	--		--	--		
Feed (calculated)	986.0	0.037	--	1.24	0.14	--	4.64	100.0	100.0
Final Preg Soln	1013.7	--	0.85	0.92	--	2.76	2.99	73.9	64.3
Leached Residue	986.0	0.010	--	0.32	0.05	--	1.66	26.1	35.7
Preg Soln, hr									
0	1014.0	--	0.00	0.00	--	0.00	0.00	0.0	0.0
2	1018.7	--	0.58	0.59	--	1.99	2.03	47.6	43.7
4	1016.6	--	0.65	0.67	--	2.45	2.52	53.9	54.3
8	1026.2	--	0.77	0.81	--	2.62	2.76	65.2	59.4
24	999.3	--	0.84	0.87	--	2.74	2.85	70.1	61.3
48	1015.1	--	0.87	0.93	--	2.73	2.92	74.6	62.9
72	1013.7	--	0.85	0.92	--	2.76	2.99	73.9	64.3

(1) Cumulative mg accounts for mg removed in sampling.

Cyanide Leaching Test 5
Sample: SAMPLE M, -65M GRIND

Results:
Reagent Consumption
NaCN, lb/ton 1.5
$Ca(OH)_2$, lb/ton 6.1

		Analysis						Distribution	
		Au			Ag				
Product	Weight g	oz/ton	ppm	mg(1)	oz/ton	ppm	mg(1)	Au %	Ag %
Feed (analyzed)			--	--		--	--		
Feed (calculated)	985.4	0.036	--	1.23	0.16	--	5.49	100.0	100.0
Final Preg Soln	1014.9	--	0.86	0.93	--	2.84	3.09	75.6	56.3
Leached Residue	985.4	0.009	--	0.30	0.07	--	2.40	24.4	43.7
Preg Soln, hr									
0	1014.6	--	0.00	0.00	--	0.00	0.00	0.0	0.0
2	1017.5	--	0.63	0.64	--	2.13	2.17	52.0	39.5
4	1019.5	--	0.66	0.68	--	2.54	2.63	55.5	47.9
8	1020.6	--	0.73	0.77	--	2.67	2.81	62.2	51.1
24	1020.7	--	0.88	0.93	--	2.80	2.98	75.7	54.4
48	1015.7	--	0.87	0.93	--	2.67	3.08	75.5	56.1
72	1014.9	--	0.86	0.93	--	2.84	3.09	75.6	56.3

(1) Cumulative mg accounts for mg removed in sampling.

Cyanide Leaching Test 6
Sample: SAMPLE M, -100M GRIND

Results:
Reagent Consumption
NaCN, lb/ton 2.34
Ca(OH)2, lb/ton 6.3

Product	Weight g	Au oz/ton	Au ppm	Au mg(1)	Ag oz/ton	Ag ppm	Ag mg(1)	Distribution Au %	Distribution Ag %
Feed (analyzed)			--	--		--	--		
Feed (calculated)	983.6	0.041	--	1.38	0.15	--	4.89	100.0	100.0
Final Preg Soln	1014.9	--	1.04	1.12	--	2.96	3.20	80.7	65.5
Leached Residue	983.6	0.008	--	0.27	0.05	--	1.69	19.3	34.5
Preg Soln, hr									
0	1016.4	--	0.00	0.00	--	0.00	0.00	0.0	0.0
2	1017.5	--	0.33	0.34	--	1.60	1.63	24.3	33.3
4	1022.9	--	0.65	0.67	--	2.41	2.49	48.5	51.0
8	1016.6	--	0.87	0.90	--	2.80	2.91	65.0	59.5
24	1009.0	--	1.04	1.08	--	2.96	3.09	77.9	63.3
48	1015.2	--	1.03	1.09	--	2.93	3.13	78.9	64.0
72	1014.9	--	1.04	1.12	--	2.96	3.20	80.7	65.5

(1) Cumulative mg accounts for mg removed in sampling.

Cyanide Leaching Test 7
Sample: SAMPLE 0, -35M GRIND

Results:
Reagent Consumption
NaCN, lb/ton 2.44
Ca(OH)2, lb/ton 4.4

Product	Weight g	Analysis Au oz/ton	Au ppm	Au mg(1)	Ag oz/ton	Ag ppm	Ag mg(1)	Distribution Au %	Distribution Ag %
Feed (analyzed)			--	--		--	--		
Feed (calculated)	985.6	0.044	--	1.49	0.06	--	2.13	100.0	100.0
Final Preg Soln	1013.0	--	1.08	1.17	--	0.40	0.44	78.7	20.5
Leached Residue	985.6	0.009	--	0.32	0.05	--	1.69	21.3	79.5
Preg Soln, hr									
0	1014.4	--	0.00	0.00	--	0.00	0.00	0.0	0.0
2	1022.2	--	0.83	0.90	--	0.38	0.39	60.4	18.3
4	1017.9	--	1.00	1.03	--	0.40	0.41	69.2	19.4
8	1014.8	--	1.08	1.13	--	0.40	0.42	75.6	19.7
24	1013.4	--	1.07	1.13	--	0.41	0.43	76.0	20.5
48	1012.6	--	1.08	1.16	--	0.40	0.43	77.6	20.2
72	1013.0	--	1.08	1.17	--	0.40	0.44	78.7	20.5

(1) Cumulative mg accounts for mg removed in sampling.

Cyanide Leaching Test 8
Sample: SAMPLE 0, -65M GRIND

Results:
Reagent Consumption
NaCN, lb/ton 2.6
Ca(OH)2, lb/ton 4.4

Product	Weight g	Au oz/ton	Au ppm	Au mg(1)	Ag oz/ton	Ag ppm	Ag mg(1)	Distribution Au %	Distribution Ag %
Feed (analyzed)			--	--		--	--		
Feed (calculated)	986.8	0.044	--	1.50	0.07	--	2.47	100.0	100.0
Final Preg Soln	1013.9	--	1.12	1.22	--	0.50	0.54	80.9	22.0
Leached Residue	986.8	0.009	--	0.29	0.06	--	1.93	19.1	78.0
Preg Soln, hr									
0	1013.2	--	0.00	0.00	--	0.00	0.00	0.0	0.0
2	1016.9	--	0.87	0.88	--	0.47	0.48	58.8	19.3
4	1015.2	--	1.03	1.06	--	0.49	0.50	70.4	20.4
8	1016.3	--	1.09	1.14	--	0.48	0.50	75.6	20.3
24	1011.7	--	1.15	1.21	--	0.49	0.52	80.6	21.0
48	1012.9	--	1.12	1.20	--	0.49	0.53	79.7	21.3
72	1013.9	--	1.12	1.22	--	0.50	0.54	80.9	22.0

(1) Cumulative mg accounts for mg removed in sampling.

Cyanide Leaching Test 9
Sample: SAMPLE O, -100M GRIND

Results:
Reagent Consumption
NaCN, lb/ton 2.7
Ca(OH)2, lb/ton 4.4

Product	Weight g	Au oz/ton	Au ppm	Au mg(1)	Ag oz/ton	Ag ppm	Ag mg(1)	Distribution Au %	Distribution Ag %
Feed (analyzed)			--	--		--	--		
Feed (calculated)	982.8	0.044	--	1.50	0.05	--	1.60	100.0	100.0
Final Preg Soln	1015.1	--	1.13	1.23	--	0.54	0.59	82.4	36.9
Leached Residue	982.8	0.008	--	0.26	0.03	--	1.01	17.6	63.1
Preg Soln, hr									
0	1017.2	--	0.00	0.00	--	0.00	0.00	0.0	0.0
2	1020.5	--	0.94	0.96	--	0.52	0.53	64.1	33.1
4	1016.8	--	1.06	1.09	--	0.54	0.56	73.2	34.8
8	1015.7	--	1.10	1.15	--	0.54	0.57	77.0	35.4
24	1015.8	--	1.13	1.20	--	0.54	0.58	80.1	35.9
48	1014.4	--	1.12	1.21	--	0.54	0.58	80.6	36.4
72	1015.1	--	1.13	1.23	--	0.54	0.59	82.4	36.9

(1) Cumulative mg accounts for mg removed in sampling.

Cyanide Leaching Test 10
Sample: SAMPLE S FLOTATION CONC, -65M GRIND (ROASTED)

Results:
Reagent Consumption
NaCN, lb/ton 0 (1)
Ca(OH)2, lb/ton 0 (2)

Product	Weight g	Au oz/ton	Au ppm	Au mg(1)	Ag oz/ton	Ag ppm	Ag mg(1)	Distribution Au %	Distribution Ag %
Feed (analyzed)			--	--		--	--		
Feed (calculated)	114.2	0.292	--	1.14	0.94	--	3.67	100.0	100.0
Final Preg Soln	244.4	--	2.99	1.03	--	4.95	1.60	89.7	43.6
Leached Residue	114.2	0.030	--	0.12	0.53	--	2.07	10.3	56.4
Preg Soln, hr									
0	272.7	--	0.00	0.00	--	0.00	0.00	0.0	0.0
2	274.1	--	3.70	1.01	--	4.27	1.17	88.6	31.9
4	257.7	--	3.70	1.02	--	4.36	1.20	89.1	32.7
8	250.6	--	3.60	1.02	--	4.61	1.29	88.8	35.1
24	255.6	--	3.40	1.03	--	4.72	1.39	89.7	38.0
48	233.9	--	3.40	1.03	--	5.19	1.51	89.9	41.1
72	244.4	--	2.99	1.03	--	4.95	1.60	89.7	43.6

1 Difficulty in free NaCN titrations prevented accurate consumption measurement.

2 Roasted sample slurry was neutralized with $Ca(OH)_2$ prior to leaching (246 lb/ton). After filtration, the solids were repulped for leaching. No additional lime was used in the leach.

Cyanide Leaching Test 11
Sample: SAMPLE 5 FLOTATION CONC. -65M GRIND (NOT ROASTED)

Results:
Reagent Consumption
NaCN, lb/ton 21.5
Ca(OH)2, lb/ton 0(1)

Product	Weight	Analysis						Distribution	
		Au			Ag			Au	Ag
	g	oz/ton	ppm	mg(1)	oz/ton	ppm	mg(1)	%	%
Feed (analyzed)			--	--		--	--		
Feed (calculated)	152.0	0.222	--	1.16	0.78	--	4.08	100.0	100.0
Final Preg Soln	306.8	--	2.19	0.67	--	5.60	2.17	76.6	53.2
Leached Residue	152.0	0.052	--	0.27	0.37	--	1.91	23.4	46.8
Preg Soln, hr									
0	363.7	--	0.00	0.00	--	0.00	0.00	0.0	0.0
2	369.0	--	2.40	0.89	--	4.51	1.66	76.5	40.8
4	351.8	--	2.36	0.87	--	4.40	1.62	75.0	39.7
8	344.9	--	2.33	0.88	--	4.49	1.69	75.8	41.4
24	339.4	--	2.15	0.85	--	4.79	1.85	73.2	45.3
48	322.9	--	2.03	0.62	--	5.07	1.97	71.0	48.3
72	306.8	--	2.19	0.89	--	5.60	2.17	76.6	53.2

1 Slurry sample was neutralized with $Ca(OH)_2$ prior to leaching, 20 lb/ton. After filtration, the solids were repulped for leaching. No additional lime was used in the leach.

EXHIBIT 4

GRINDABILITY TESTS

Grindability Test 1

Purpose: To determine the ball mill grindability of the test sample in terms of a Bond work index number.

Sample: Oxidized ore crushed to -6M.

Procedure: The equipment and procedure duplicate the Bond method for determining ball mill work indices.

Test Conditions: Mesh of grind: 65
Weight of undersize product for 250% circulating load: 309.1 g
Weight % of undersize material in ball mill feed: 17.51

Results:

Stage No.	New Feed g	Undersize In Feed g	Undersize To Be Ground g	Revolutions	Undersize in Product g	Undersize Produced Total g	Undersize Produced Per Mill Revolution g
1	1,082.0	189.5	119.6	40	253.8	64.3	1.608
2	253.8	44.4	264.7	165	321.3	276.9	1.678
3	321.3	56.3	252.8	151	309.2	252.9	1.675
4	309.2	54.1	255.0	152	298.1	244.0	1.605
5	298.1	52.2	256.9	160	332.5	280.3	1.752
6	332.5	58.2	250.9	143	327.2	269.0	1.881
7	327.2	57.3	251.8	134	310.8	253.5	1.892
8	310.8	54.4	254.7	135	318.5	264.1	1.956
10	308.5	54.0	255.1	130	313.8	259.8	1.998
11	313.8	54.9	254.2	127	310.9	256.0	2.016
12	310.9	54.4	254.7	126	--	--	1.974

Average Last Three = 1.996

Ball Mill Work Index Computations

$$Wi = \frac{44.5}{P_1^{0.23} \times Gbp^{0.82} \times \left(\frac{10}{\sqrt{P}} - \frac{10}{\sqrt{F}}\right)}$$

Wherein: P_1 = 100% Passing Size of Product = 212 µm
Gbp = Grams per Revolution = 1.996
P = 80% Passing Size of Product = 165 µm
F = 80% Passing Size of Feed = 2,600 µm

Wi = 12.6

EXHIBIT 4

Grindability Test 1 -- continued

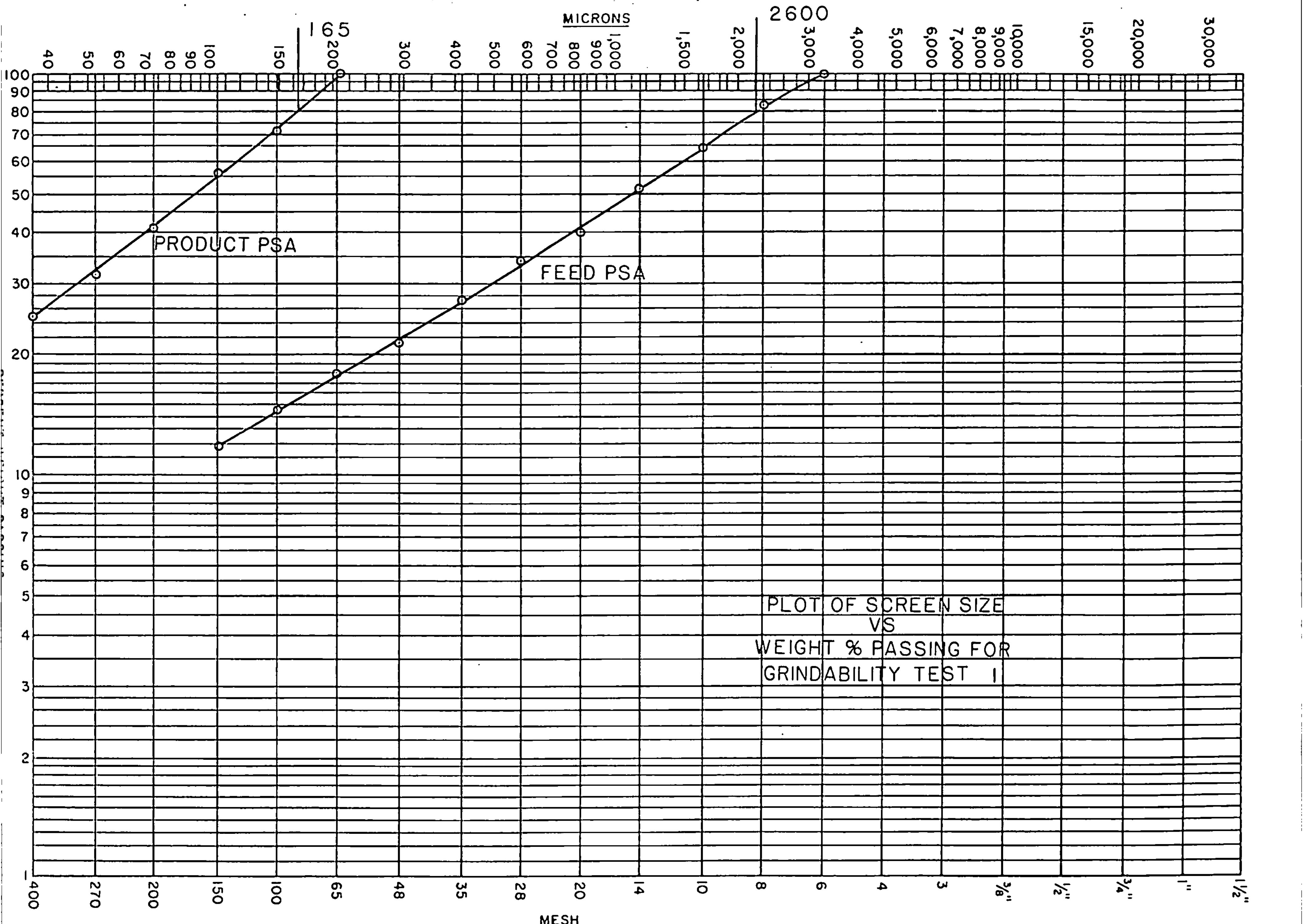
Feed Particle Size Analyses

Direct: Screen Product (Tyler) Mesh		Direct: Weight %	Cumulative Passing: (Tyler) Mesh	Cumulative Passing: Weight %
Head (calculated)		100.00		
	+8	17.63	6	100.00
-8	+10	18.52	8	82.37
-10	+14	13.39	10	63.85
-14	+20	10.78	14	50.46
-20	+28	6.52	20	39.68
-28	+35	6.74	28	33.16
-35	+48	5.14	35	26.42
-48	+65	3.77	48	21.28
-65	+100	3.18	65	17.51
-100	+150	2.79	100	14.33
-150		11.54	150	11.54

Product Particle Size Analysis[1]

Direct: Screen Product (Tyler) Mesh		Direct: Weight %	Cumulative Passing: (Tyler) Mesh	Cumulative Passing: Weight %
Head (calculated)		100.00		
	+100	28.32	65	100.00
-100	+150	15.70	100	71.68
-150	+200	15.13	150	55.98
-200	+270	8.78	200	40.85
-270	+400	6.96	270	32.07
-400		25.11	400	25.11

1 -65M product combined from Stages 10, 11, and 12 of Grindability Test 1.



MICRONS
165
2600
PRODUCT PSA
FEED PSA
MESH
PLOT OF SCREEN SIZE
VS
WEIGHT % PASSING FOR
GRINDABILITY TEST 1

EXHIBIT 4

Grindability Test 2

Purpose: To determine the ball mill grindability of the test sample in terms of a Bond work index number.

Sample: Mixed sulfide and oxide ore crushed to -6M.

Procedure: The equipment and procedure duplicate the Bond method for determining ball mill work indices.

Test Conditions: Mesh of grind: 65
Weight of undersize product for 250% circulating load: 337.1 g
Weight % of undersize material in ball mill feed: 33.88

Results:

Stage No.	New Feed g	Undersize In Feed g	Undersize To Be Ground g	Revolutions	Undersize in Product g	Undersize Produced Total g	Undersize Produced Per Mill Revolution g
1	1,179.8	399.7	62.6	0	399.7	--	--
2	399.7	135.4	201.7	44	272.0	136.6	3.105
3	272.0	92.2	244.9	79	281.3	189.1	2.394
4	281.3	95.3	241.8	101	311.3	216.0	2.139
5	311.3	105.5	231.6	108	332.3	226.8	2.100
6	332.3	112.6	224.5	107	321.9	209.3	1.956
7	321.9	109.1	228.0	117	365.4	256.3	2.191
8	365.4	123.8	213.3	97	337.9	214.1	2.207
9	337.9	114.5	222.6	101	339.7	225.2	2.229
10	339.7	115.1	222.0	100	344.7	229.6	2.296
11	344.7	116.8	220.3	96	330.6	213.8	2.227

Average Last Three = 2.251

Ball Mill Work Index Computations

$$Wi = \frac{44.5}{P_1^{0.23} \times Gbp^{0.82} \times \left(\frac{10}{\sqrt{P}} - \frac{10}{\sqrt{F}} \right)}$$

Wherein: P_1 = 100% Passing Size of Product = 212 µm
Gbp = Grams per Revolution = 2.251
P = 80% Passing Size of Product = 155 µm
F = 80% Passing Size of Feed = 1,280 µm

Wi = 12.7

EXHIBIT 4

Grindability Test 2 -- continued

Feed Particle Size Analyses

Direct: Screen Product (Tyler) Mesh	Direct: Weight %	Cumulative Passing: (Tyler) Mesh	Cumulative Passing: Weight %
Head (calculated)	100.00		
+8	6.51	6	100.00
-8 +10	7.94	8	93.49
-10 +14	7.84	10	85.55
-14 +20	9.40	14	77.71
-20 +28	8.30	20	68.31
-28 +35	9.27	28	60.01
-35 +48	8.83	35	50.74
-48 +65	8.03	48	41.91
-65 +100	7.05	65	33.88
-100 +150	6.70	100	26.83
-150	20.13	150	20.13

Product Particle Size Analysis[1]

Direct: Screen Product (Tyler) Mesh	Direct: Weight %	Cumulative Passing: (Tyler) Mesh	Cumulative Passing: Weight %
Head (calculated)	100.00		
+100	23.68	65	100.00
-100 +150	18.94	100	76.30
-150 +200	12.48	150	57.38
-200 +270	9.52	200	44.90
-270 +400	8.64	270	35.38
-400	26.74	400	26.74

1 -65M product combined from Stages 9, 10, and 11 of Grindability Test 2.

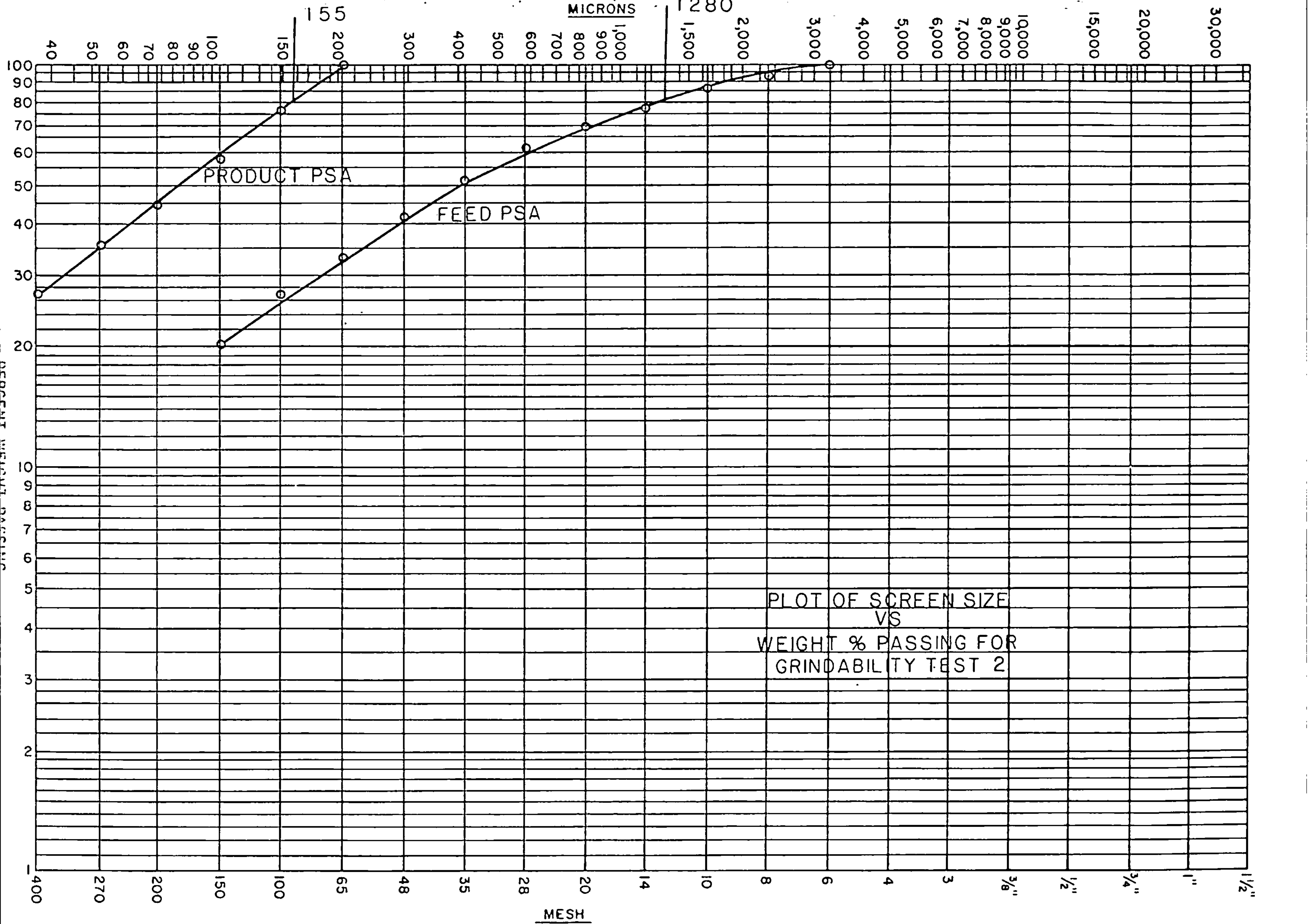

MICRONS
155
1280
PRODUCT PSA
FEED PSA
PLOT OF SCREEN SIZE
VS
WEIGHT % PASSING FOR
GRINDABILITY TEST 2
MESH

EXHIBIT 4

Grindability Test 3

Purpose: To determine the ball mill grindability of the test sample in terms of a Bond work index number.

Sample: Sulfide ore crushed to -6M.

Procedure: The equipment and procedure duplicate the Bond method for determining ball mill work indices.

Test Conditions: Mesh of grind: 65
Weight of undersize product for 250% circulating load: 344.5 g
Weight % of undersize material in ball mill feed: 33.09

Results:

Stage No.	New Feed g	Undersize In Feed g	Undersize To Be Ground g	Revolutions	Undersize in Product g	Undersize Produced Total g	Undersize Produced Per Mill Revolution g
1	1,205.6	398.9	54.4	0	398.9	--	--
2	398.9	132.0	212.5	42	257.6	125.6	2.990
3	257.6	85.2	259.3	87	258.4	173.2	1.991
4	258.4	85.5	259.0	130	314.7	229.2	1.763
5	314.7	104.1	240.4	136	350.8	246.7	1.814
6	350.8	116.1	228.4	126	349.2	233.1	1.850
7	349.2	115.5	229.0	124	365.9	250.1	2.019
8	365.9	121.1	223.4	111	352.3	231.2	2.083
9	352.3	116.6	227.9	109	342.8	226.2	2.075

Average Last Three = 2.059

Ball Mill Work Index Computations

$$Wi = \frac{44.5}{P_1^{0.23} \times Gbp^{0.82} \times \left(\frac{10}{\sqrt{P}} - \frac{10}{\sqrt{F}}\right)}$$

Wherein: P_1 = 100% Passing Size of Product = 212 μm
Gbp = Grams per Revolution = 2.059
P = 80% Passing Size of Product = 161 μm
F = 80% Passing Size of Feed = 1,460 μm

Wi = 13.6

EXHIBIT 4

Grindability Test 3 -- continued

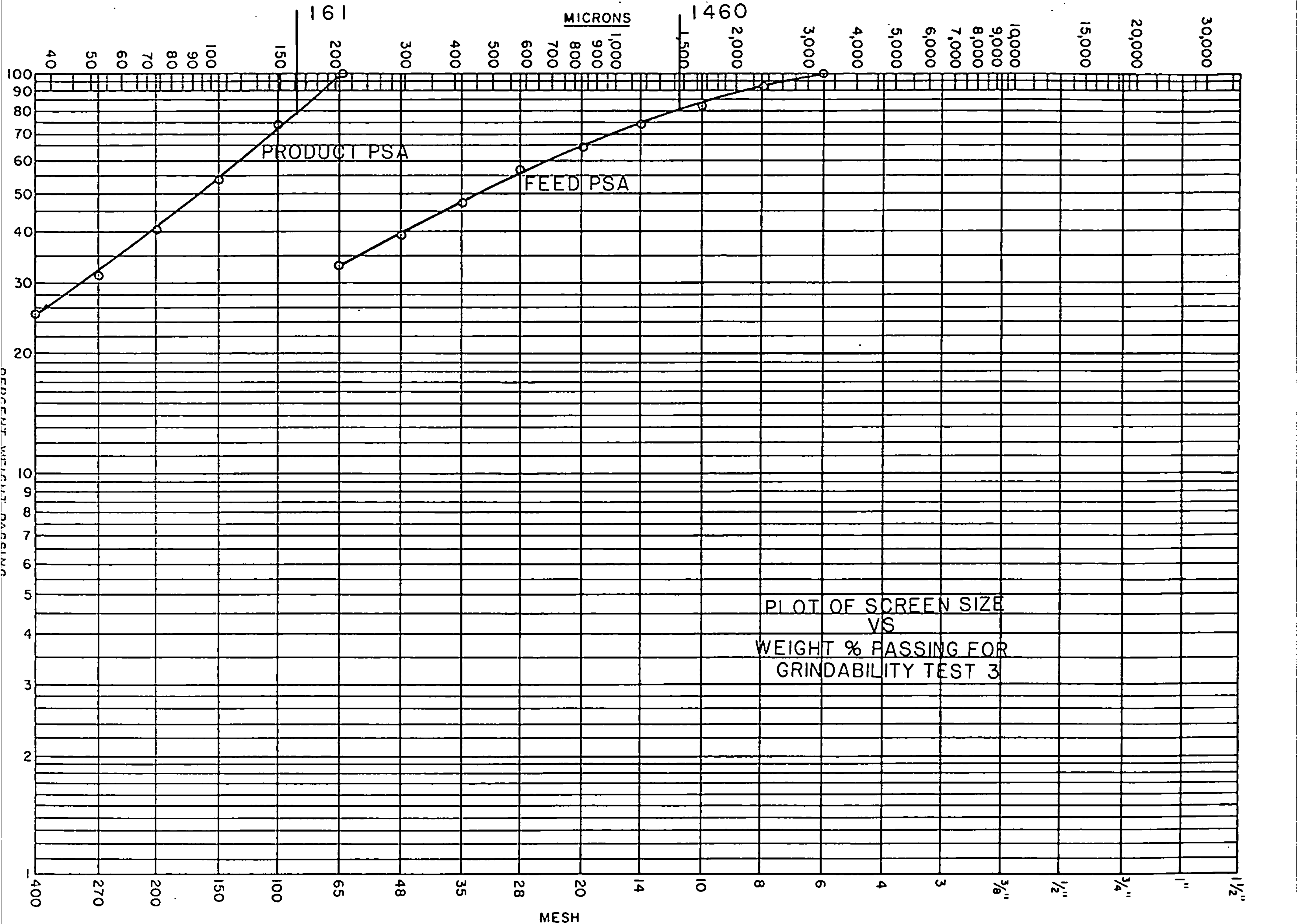
Feed Particle Size Analyses

Direct: Screen Product (Tyler) Mesh	Direct: Weight %	Cumulative Passing: (Tyler) Mesh	Cumulative Passing: Weight %
Head (calculated)	100.00		
+8	6.92	6	100.00
-8 +10	9.25	8	93.08
-10 +14	9.18	10	83.83
-14 +20	10.53	14	74.65
-20 +28	8.61	20	64.12
-28 +35	7.88	28	55.51
-35 +48	7.77	35	47.63
-48 +65	6.77	48	39.86
-65 +100	5.87	65	33.09
-100 +150	5.48	100	27.22
-150	21.74	150	21.74

Product Particle Size Analysis[1]

Direct: Screen Product (Tyler) Mesh	Direct: Weight %	Cumulative Passing: (Tyler) Mesh	Cumulative Passing: Weight %
Head (calculated)	100.00		
+100	25.86	65	100.00
-100 +150	19.61	100	74.14
-150 +200	13.39	150	54.53
-200 +270	9.54	200	41.14
-270 +400	5.45	270	31.60
-400	26.15	400	26.15

[1] -65M product combined from Stages 7, 8, and 9 of Grindability Test 3.



161
MICRONS
1460
PRODUCT PSA
FEED PSA
PLOT OF SCREEN SIZE
VS
WEIGHT % PASSING FOR
GRINDABILITY TEST 3
MESH

EXHIBIT 4

Grindability Test 4

Purpose: To determine the rod mill grindability of the test sample in terms of a Bond work index number.

Sample: Oxidized ore crushed to -½ in.

Procedure: The equipment and procedure duplicate the Bond method for determining ball mill work indices.

Test Conditions: Mesh of grind: 14
Weight of undersize product for 250% circulating load: 936.2 g
Weight % of undersize material in ball mill feed: 27.58

Results:

Stage No.	New Feed g	Undersize In Feed g	Undersize To Be Ground g	Revolutions	Undersize in Product g	Undersize Produced Total g	Undersize Produced Per Mill Revolution g
1	1,872.4	516.4	419.8	12	651.7	135.3	11.275
2	651.7	179.7	756.5	67	955.3	775.6	11.567
3	955.3	263.4	672.8	58	983.1	719.7	12.409
4	983.1	271.1	665.1	54	960.1	689.0	12.759
5	960.1	264.8	671.4	53	977.7	712.9	13.450
6	977.7	269.6	666.6	50	951.0	681.4	13.628
7	951.0	262.3	673.9	49	910.4	648.1	13.226
						Average Last Three =	13.435

Rod Mill Work Index Computations

$$Wi = \frac{62}{P_1^{0.23} \times Gbp^{0.625} \times \left(\frac{10}{\sqrt{P}} - \frac{10}{\sqrt{F}}\right)}$$

Wherein:
P_1 = 100% Passing Size of Product = 1,168 μm
Gbp = Grams per Revolution = 13.435
P = 80% Passing Size of Product = 890 μm
F = 80% Passing Size of Feed = 7,850 μm

Wi = 10.8

EXHIBIT 4

Grindability Test 4 -- continued

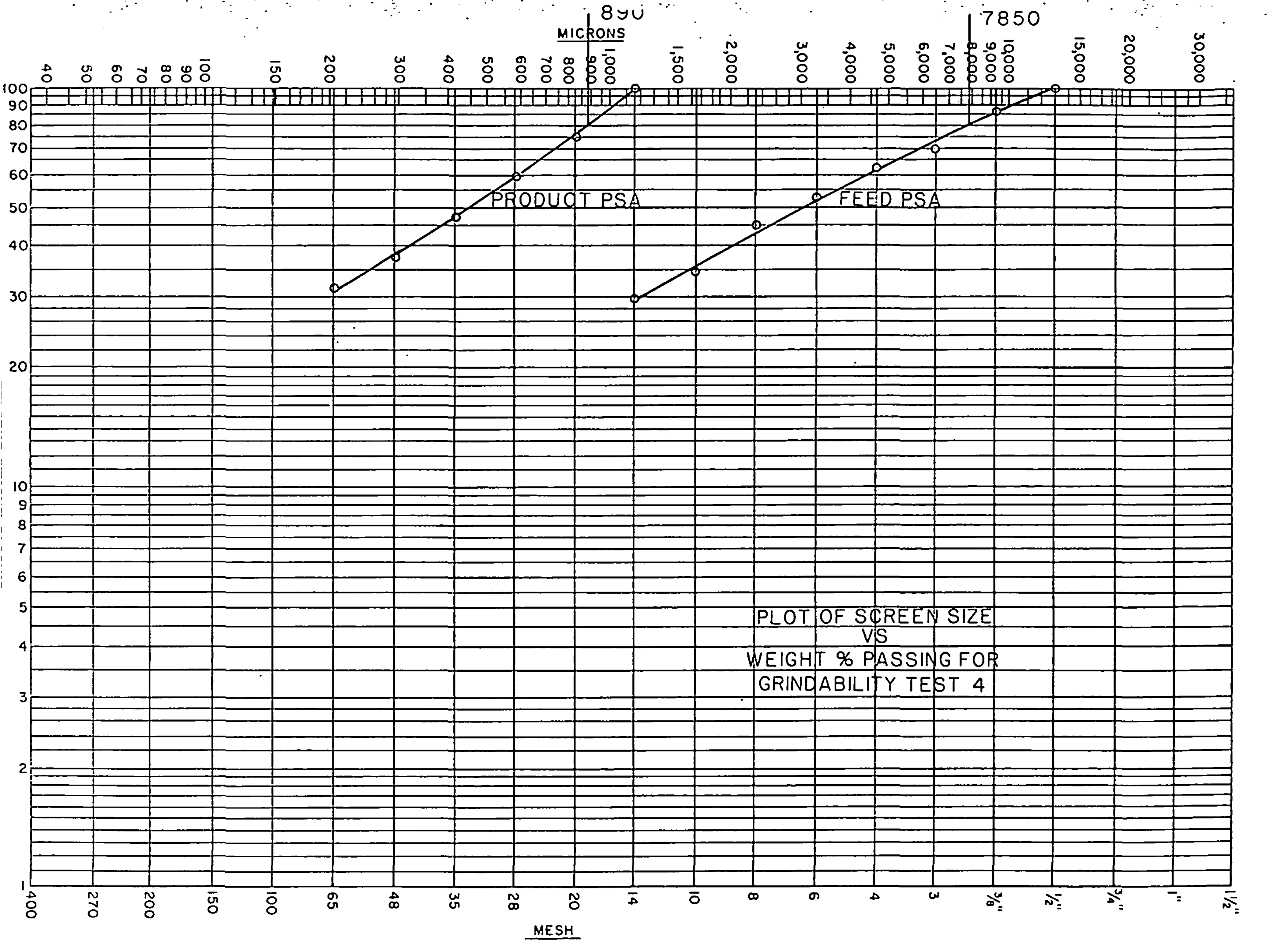
Feed Particle Size Analyses

Direct			Cumulative Passing	
Screen Product (Tyler) Mesh		Weight %	(Tyler) Mesh	Weight %
Head (calculated)		100.00		
	+3/8 in.	14.67	1/2 in.	100.00
-3/8 in.	+3M	17.95	3/8 in.	85.33
-3	+4	11.62	3M	67.37
-4	+6	8.91	4	55.76
-6	+8	7.67	6	46.85
-8	+10	6.08	8	39.18
-10	+14	5.52	10	33.09
-14		27.58	14	27.58

Product Particle Size Analysis[1]

Direct			Cumulative Passing	
Screen Product (Tyler) Mesh		Weight %	(Tyler) Mesh	Weight %
Head (calculated)		100.00		
	+20	25.20	14	100.00
-20	+28	15.28	20	74.80
-28	+35	12.88	28	59.52
-35	+48	9.48	35	46.64
-48	+65	6.73	48	37.16
-65		30.43	65	30.43

1 -14M product combined from Stages 5, 6, and 7 of Grindability Test 4.



890
7850
MICRONS
40 50 60 70 80 90 100 150 200 300 400 500 600 700 800 900 1,000 1,500 2,000 3,000 4,000 5,000 6,000 7,000 8,000 9,000 10,000 15,000 20,000 30,000
100 90 80 70 60 50 40 30 20 10 9 8 7 6 5 4 3 2 1
PRODUCT PSA
FEED PSA
PLOT OF SCREEN SIZE
VS
WEIGHT % PASSING FOR
GRINDABILITY TEST 4
400 270 200 150 100 65 48 35 28 20 14 10 8 6 4 3 3/8" 1/2" 3/4" 1" 1 1/2"
MESH

J. BARRON

CSMRI SPLITS
FOR MET TESTING

Selected Holes for Metallurgical Testing.

1 RGE 37 - Sulphides, Mixed and Oxides in rock type 20;
Entire hole lies just outside Oxide pit area
Assays consistently range from .08, .07, .06, .05, .03 .02 ~.015;

2 RGE 38. a) 150-2?? Sulphides, rock type 12, Assays ~.04
hole located in Dakota Maid area.
b) 150-270 Sulphides, rock type 20, Assays ~ .15 - ~.02
c) 270-305 Mixed, rock type 20 Assays ~.15 - ~.02

3. GLE 28 200-210 Oxide, rock type 20 Assay .027
located south west of Pit

4. GLE 32 250-310 Sulphide rock type 30 Assay ~.016
located south of pits (Langley Stock)

5 GLE 44 a) 10 - 30 Oxide rock type 30 Assay ~0.12
located south of pits (Langley Stock)

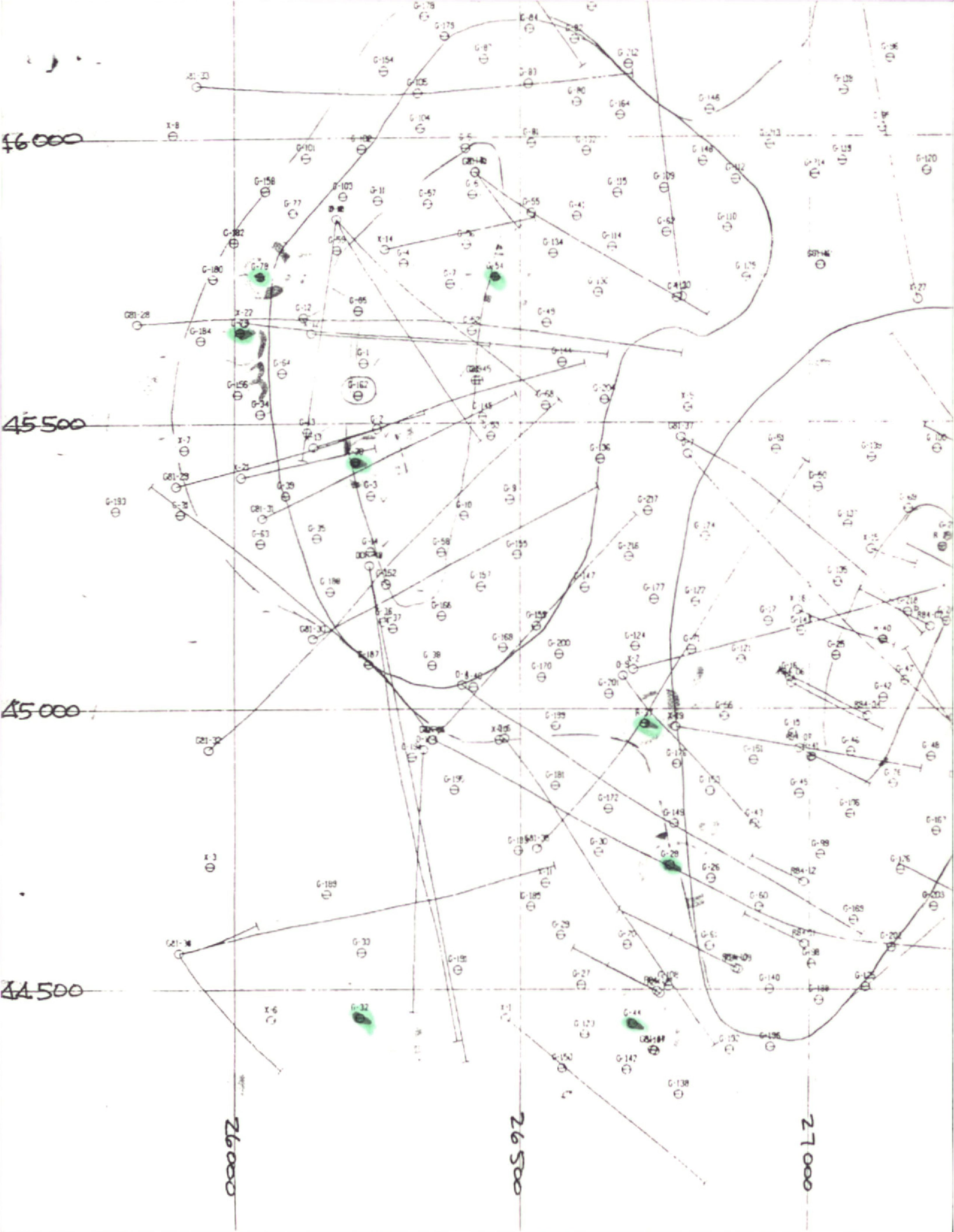
b) 210-260 Sulphide rock type 30 Assay ~.05
located south of pits (Langley Stock).

6 GLE 54 180-210 Mixed, rock type 30 Assay ~.02
bottom of Dakota Maid Pit.

7 GLE 78 0 - 60 Oxide rock type 10 Assay ~ .02
west edge of Dakota Maid Pit.

8 GE 79 90-100. Mixed, rock type 10 Assay .021
west edge of Dakota Maid Pit

Min. Type \ Rock Type	10	20	30
1	2(a)	1, 2(b)	4, 5(b).
2	8	1, 2(c)	6
3	7	1, 3	5 (a).



46000
45500
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44500
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27000

LACANA MINING CORPORATION
Suite 3701, Royal Trust Tower
Box 354, Toronto-Dominion Centre
Toronto, Ontario, Canada M5K 1K7
416-367-0840 Telex: 06-218157

February 8, 1984

Mr. R.P. Hackl
Extractive Metallurgist
B.C. Research
3650 Westbrook Mall
Vancouver, B.C. V6S 2L2

Dear Mr. Hackl:

Thank you for your letter of February 3rd.

Please proceed on the revised program as outlined in your letter. I will ask our Coeur d'Alene office to ship another 30 lbs.

I am sending a copy of this letter to Dr. Ric Lawrence, as I am not sure when your absence from the office begins.

Best regards.

Yours very truly

LACANA MINING CORPORATION

E.G. Thompson
President and
Chief Executive Officer

cc: R. Lawrence
Coeur d'Alene Office
Reno Office

MAR FEB - 6 1984

B.C. RESEARCH

3650 Wesbrook Mall, Vancouver, Canada V6S 2L2

Phone (604) 224-4331 • Cable 'RESEARCHBC' • Telex 04-507748

February 3, 1984
Our File: 1-41-571

Mr. E.G. Thompson
President and Chief Executive Officer
Lacana Mining Corporation
P.O. Box 354
Suite 3701 Royal Trust Tower
T-D Centre
Toronto, Ontario
M5K 1K7

Dear Mr. Thompson:

Re: Revised Program to Evaluate Biological Pre-oxidation of Gilt Edge Ore

Further to our meeting last Tuesday, January 31, we have prepared a revised proposal for your consideration.

Preliminary test work has shown that 85% gold recovery by straight cyanidation is possible from finely milled Gilt Edge ore. It is not clear to what extent milling liberates gold from associated pyrite, but it appears to be significant and therefore biological preoxidation tests are not really justified.

Because a heap leaching operation is being considered for this ore, we feel that the best way to assess the viability of a biological pre-oxidation step is to carry out small column leach tests on coarser material, say -1/4" or -1/8". One column test would be a straight cyanide leach to determine rate and extent of gold recovery possible from untreated material. Biological leaching would be initiated in two other columns with the idea of leaching to two different degrees of pyrite breakdown, ie. 25% and 60%+. If gold recovery by cyanidation is high, the two biological leaching columns can be terminated at any time. However, if the untreated ore does prove to be refractory to cyanidation, the pre-oxidized columns can then be cyanided to determine the extent of improved gold recovery possible, and a rough idea of the degree of pyrite oxidation required for improved gold recovery.

Leaching would be carried out on 13 lb. samples in our 32" long by 3" diameter columns. At present we have only enough as-received sample for one column, so we would require another 30 lbs. The estimated cost breakdown and time required are as follows.

...../ 2

Test	Duration (weeks)	Cost $
1 CN Column Test	6	2,000.00
2 Biological/Cyanide Column Tests, if required	16	4,500.00
		$6,500.00

The above cost includes all material handling, analytical, supervision and reporting charges. The approximate expenditure for work performed to date is $2,500.00 out of a $5,000.00 budget; therefore we would require an additional $4,000.00 if all of the above work is carried out.

The columns could be started within 2 weeks of receiving your approval and additional sample.

Ab Bruynesteyn and myself will be away until February 20 and March 12 respectively, but Dr. Ric Lawrence has been fully briefed on this project and would be pleased to answer any questions.

Sincerely yours,

B. C. RESEARCH

R P Hackl

R.P. Hackl
Extractive Metallurgist
Division of Extractive Metallurgy

RPH/jn

D. M. DUNCAN, INC.

MINING DEVELOPMENT · MANAGEMENT

2555 Sharon Way
Reno, Nevada 89509
Telephone 702-826-0890

December 20, 1982

Mr. Paul E. Dircksen
Lacana Mining Incorporated
2005 Ironwood Parkway, Room 105
Coeur d'Alene, Idaho 83814

Dear Paul:

Recently you provided me with copies of five metallurgical reports on the Gilt Edge property in South Dakota. They were dated Nov. 10, 1981, March 19, 1982, May 12, 1982, July 6, 1982, and Aug. 2, 1982. With the exception of the Cyprus report dated May 12, 1982, the data was the work of Dan Kappes. The work is summarized as follows:

1. The 1981 report by Kappes discusses the results of 12 bucket leach tests on ore from various parts of the property. Extractions range between 48% and 84% with an arithmetic average of 63%.

2. The March 19, 1982 report discusses the results of approximately 500 Kappes style leach tests on both pulverized and non-pulverized drill hole samples. I believe the extractions average about 75% of pulverized material and 66% on non-pulverized. I would emphasize here that these extractions are arithmetic averages and may be quite different from averages weighted by ore types. Also, they represent only gold taken into solution and do not account for soluable losses such as a milling operation incurs.

3. The work done by Dobson of Cyprus reports on 200 gram agitated leach tests of ore at various grinds, some flotation work followed by leaching of concentrates and also some leaching of roasted float con. The work does not detail sample types except by an alphabetical letter. All the leach tests suggest that a grind of 65% minus 200 mesh is about optimum. Data reported is very erratic and we assume it is the result of course gold. It suggests larger samples are needed and possibly special procedures such as pre concentration of the heavy fraction. I would not place too much emphasis on this work.

Flotation work on a composite sample provided a gold recovery of 85% in a concentrate with no specified ratio of concentration. Subsequent leaching of the con recovers 83% of the contained gold. Overall recovery, accordingly, is 70%.

Roasting of the con followed by leaching recovered 97% of the contained gold, for an overall 82%. Optimization work would no doubt improve these numbers.

Results of the flotation work suggests gravity concentration should be attempted.

4. The report dated July 6, 1982 by Dan Kappes attempts to summarize all metallurgical testing. He states that the testing indicated recoveries of 70% for crushed (minus 2") oxidized ore and that the Sunday ore performed better than the Dakota Maid. Suggested recovery for sulfide ore was highly variable and averaged 45-50 percent, again on minus 2" or finer. Potential for recovery in a cyanide mill is stated to be 76%. He refers to a historic gold recovery of 75% attained during the 1930's. There is no mention of the flowsheet (type of mill).

5. Report by Kappes dated Aug. 2, 1982 concerns the four 40' high column tests. Average extraction for 3 columns (normal ore) was 75%. Leaching times ranged between 80 and 210 days. Extraction on the 25% sulfide ore contained in column 4 was 82% in only 70 days with a good ongoing rate of recovery (as shown on graph). This latter is quite anomalous, particularly when compared with the corresponding bucket leach test (50% extraction). Results on the column tests (with exception of column 4) compare well with bucket tests. Extraction times are noted by Kappes and should be indicative of 40' high heaps. Cyanide consumption for the tests averaged 1.5#. This could have been reduced significantly if the ore had been neutralized first. $Ca(OH)_2$ consumption was stated to be a remarkably low .5#.

6. The report on the 1700 ton run-of-mine leach test was not provided but results are mentioned in the Kappes July 6, 1982 Summary Report. He calculates a 46% gold extraction in 130 days during the first season, and an additional 7% in 30 days during the next season. He states that results were disappointing and were due primarily to "non-ideal" stacking procedures. It is the writer's opinion that the recoveries stated are realistic numbers and that stacking procedures have little to do with it. The leaching times were excessive and if the heaps were not neutralized prior to leaching (with strong NaOH solution) this would account for much of the problem.

Additional Comments

For the amount of information obtained there has been an excessive amount of testing done and on too small a scale. To complete the work, I would recommend two further tests, each

4,000-5,000 tons. One on run-of-mine ore and the other on ore crushed to 3/4" half of which is agglomerated. One end of the crushed ore heap would be agglomerated and the other end un-agglomerated.

Sampling of trenches cut through the heap's tailings would determine the merits, if any, of agglomerating. It might be useful to conduct the testing over a period of two seasons in order to determine the degree of compaction and it's affects on percolation, over the prolonged period.

If a carbon column recovery system is ever contemplated, it's design should make provision for recovery of at least as much silver and copper as gold.

For estimating purposes, assume Cyanide consumption at 1#/ton and lime (CaO) at 3#/ton.

Some testing of gravity concentration on both sulfide and oxidized ores should be done. I would recommend that Larry Mashburn of Boise Assay Lab do this.

Yours sincerely,

Don

D.M. Duncan

/fap

3365 South Akron Street
Denver, Colorado 80231

January 27, 1983

Mr. Paul E. Dirksen
Lacana Mining Incorporated
2005 Ironwood Parkway, Room 105
Coeur d'Alene, Idaho 83814

Dear Paul:

A brief review has been made on the Gilt Edge and Turner-Albright properties.

The Gilt Edge metallurgy is extensive, but has some unanswered questions. These answers may be found in other reports; however, I can not find explanations on wide extraction variations other than the possibility of "free gold" occurences. Geologic information available does not make reference to this being the case.

Leaching rates on the unoxidized ore is variable, but 50% extraction is the best to be expected on ores crushed to minus 2 inches. Improved extractions could be reasonably expected if an oxidant and higher cyanide concentrations were used on future test work.

Kappes reports in the Gilt Edge Final Report-Bucket Leach Tests 1970 Mini-Bulk Samples-10 November, 1981: "The data clearly indicates That the gold is concentrated into the smaller size fractions, which is an indicator that it occurs primarily on fracture surfaces within the rock."

Accepting this to be the case, a carefully sized, attritioned and agglomerated ore should be tested for a heap leaching operation. Neutralization of the ore should be done prior to beginning cyanide leaching.

The use of sodium hydroxide as a buffer should be at least reviewed. There is a reflection that lime may be interfering, based on Gilt Edge Report 1982-D, 2 August 1982, Page 38 - Test 985 and 996.

Well prepared samples should be quartered, split and assayed by fire assay on at least assay ton samples for gold and silver. Copper should be assayed on each sample prior to beginning testing.

Future testing should include assaying of all residues. Metal balances should not be calculated using recoveries in excess of 100%.

Mr. Paul E. Dirksen
January 27, 1983
Page 2

This error may be caused by not carefully measuring and assaying solutions on a timely basis or the assaying is incorrect. Failure to assay residues will also add to errors. Figure 3 - Agitated Cyanide Leach Tests, 1979 Mini-Bulk Samples, 10 November 1981 Kappes Report is an example of this problem. (See attached copy)

Specific testing should include a limited test series on composites of oxidized near surface sample. Ore should be crushed to minus 1/2" attritioned and agglomerated with a series of lime-cyanide and a caustic-cyanide series. Gold, silver and copper head assays should be taken prior to beginning these tests. The test samples should be buffered to pH 10.6 and cyanide solutions adjusted to an excess initially of gold, silver and copper values based on head assays.

The unoxidized high sulfide ore extractions could possibly be improved by the use of a strong oxidizer along with a lead salt to reduce potential soluble sulfide interference. Higher than normal cyanide consumptions may be experienced if the ore is crushed to minus 1/2".

The Turner-Albright test work is more concise and straightforward for possible metallurgical improvement. The metallurgy is complex and a longrange test program is indicated, beyond what has been done already.

The Dawson report of May-July 1982 made recommendations for additional testing which should be done Additional suggestions are:

> Test No. 13 with lime should be repeated. Modifications to the one series would be to coarsen the grind, targeting copper grades at 15 to 20% and 1-1.25 ounces of gold.
>
> Repeat Test No. 13 using soda ash in place of lime in the copper circuit. Clean copper concentrate with soda ash. Target copper grades at 15-20%.

Based on information available, neither standard flotation nor cyanidation parameters have been established. If flotation can not be successfully applied to improve grade and extractions a combination flotation and leaching of the tailings approach will be necessary.

A comprehensive testing program appears necessary to produce a marketable copper concentrate. Acceptable zinc concentrate grades are questionable. Gold recovery from tailings will be required to make the project successful.

As I stated earlier, the cost of this testing program would be about $200,000 and would offer a challenge at the same time.

I am glad to make this review and I am looking forward to working with you on the projects.

Sincerely,

W. Bruce Brogoitti

W. Bruce Brogoitti

WBB/rsb

Enclosure

FIGURE 3. AGITATED CYANIDE LEACH TESTS

ON PULVERIZED PORTIONS OF SAMPLE SIZE FRACTIONS

(oz gold per ton/percent gold recovery/gold fineness)

SAMPLE NO.	BUCKET TEST NO.	+3 mesh	-3 + 65 mesh	- 65 + 150 mesh	- 150 mesh	WEIGHTED AVERAGE
773 A	774	.072/ 81.9%/655	.042/ 66.7%/583	.062/270.9%/423	.540/107.6%/645	.072/ 86.1%/625
773 B	775	.040/ 70.0%/549	.044/120.4%/726	.078/102.6%/800	.118/111.0%/704	.044/ 90.9%/621
773 C	776	.012/ 75.0%/ 25	.010/240.0%/118	.032/115.6%/ 62	.078/ 98.7%/ 61	.013/130.8%/ 58
773 D	777	.016/ 93.7%/577	.018/161.1%/744	.088/143.2%/863	.212/106.6%/834	.023/117.4%/649
773 E	778	.003/400.0%/571 (tr)	.003/266.7%/444 (tr)	.054/ 85.2%/807	.062/103.2%/780	.005/240.0%/534
773 F	779	.028/110.7%/663	.016/150.0%/706	.068/ 73.5%/833	.122/104.1%/830	.027/114.8%/685
773 G	780	.016/ 93.7%/349	.012/133.3%/348	.066/ 57.6%/731	.106/ 74.5%/687	.018/ 94.4%/363
773 H	781	.120/ 90.8%/122	.160/ 98.1%/194	.430/ 62.3%/221	.530/ 87.4%/177	.150/ 92.0%/151
773 I	782	.012/ 75.0%/ 54	.032/ 37.5%/ 57	.036/ 72.2%/ 43	.046/ 84.8%/ 33	.021/ 52.4%/ 54
773 J	783	.068/ 67.6%/267	.074/ 55.4%/194	.352/ 81.2%/286	.524/ 66.4%/177	.082/ 67.1%/238
773 K	784	.332/ 82.7%/846	.082/ 70.7%/659	.244/ 82.8%/811	.152/ 52.6%/559	.235/ 80.8%/770
773 L	785	.016/150.0%/480	.024/120.8%/491	.076/165.8%/829	.130/109.2%/721	.023/130.4%/496
AVERAGE		.061/ 86.3%/430	.043/ 92.6%/439	.132/ 91.6%/551	.218/ 90.0%/517	.059/ 88.1%/437

DAWSON
METALLURGICAL
LABORATORIES, INC.

P.O. Box 7685
5217 Major Street
Murray, Utah 84107-0685
Phone: 801-262-0922

OCT 4 REC'D

October 2, 1985

Lacana Gold Incorporated
2005 Ironwood Parkway, Room 105
Coeur d' Alene, Idaho 83814

Attn: Mr. Richard T. Hall

Subject: Results of Cyanide Leach Amenability Testing and Assay Screen Analyses of Gilt Edge Sulfide Ore Samples. Our Project No. P-1045-L.

Gentlemen:

Pursuant to discussions with Mr. Richard T. Hall cyanide leach amenability tests were performed on a sample of Gilt Edge sulfide ore to determine if the ore, as represented by the sample received, is amenable to cyanide leaching at a relatively coarse size of minus 1 1/2 inch.

The results of these bottle roll cyanide amenability tests indicate that it would be highly unlikely that a heap leach on an ore, as represented by this sample, could be economically successful. The results of the samples tested and reported on May 8, 1984, indicate that crushing and grinding to a much finer size improves gold recovery. It is unlikely that this could be economically successful.

Summary of Results

Results of the cyanide leach tests on ore samples crushed thru 1 1/2 and 3/4 inch are summarized in the following table, and show that less than one-third of the gold was extracted in the bottle roll cyanide amenability leach tests.

Project P-1045-L
Lacana Gold
Results of Cyanide Amenability Tests

Test	Assay, oz/Ton				%		Reagent Consumption	
	Residue		Head (calc)		Extraction		lb/Ton Ore	
	Au	Ag	Au	Ag	Au	Ag	Lime	NaCN
1 (-1 1/2" crush)	0.039	0.04	0.049	0.09	20.2	55.3	1.1	2.5
4 (-1 1/2" crush)	0.035	0.21	0.043	0.5	18.9	57.7	2.0	4.6
5 (-3/4" crush)	0.040	0.12	0.060	0.44	33.3	72.7	2.0	5.4

The results of the assay screen analyses show that the minus 35 mesh fractions had gold concentrations that were much higher than the total head assays; however, the gold extracted by cyanide leaching this fraction was still only 52 to 58 percent. The results of the assay screen analyses are summarized in the following table:

P-1045-L
Lacana Gold
Results of Assay Screen Analyses

Size Fraction	Head Analysis		Leach Resi. Analysis	
	WT %	Au, oz/T	WT %	Au, oz/T
Sample Crushed to -1 1/2 Inch				
-1 1/2" +1"	16.0	0.028	16.8	0.031
-1" +3/4"	27.1	0.030	21.5	0.035
-3/4" +1/2"	17.7	0.045	16.0	0.024
-1/2" +1/4"	13.7	0.040	11.6	0.024
-1/4" +35 Mesh	18.2	0.052	16.7	0.030
-35 Mesh	7.3	0.145	17.4	0.061
Sample Crushed to -3/4"				
-3/4" +1/2"	21.0	0.060	15.1	0.032
-1/2" 1/4"	33.2	0.036	28.7	0.035
-1/4" +35 Mesh	34.7	0.051	33.5	0.035
-35 Mesh	11.1	0.122	22.7	0.058

The increase in the weight percent in the minus 35 fraction of the leach residue over the head was probably a result of attritioning in the rolling bottles during leaching.

The complete test conditions and results are given on copies of laboratory test sheets attached to this report.

Test Procedures

The sample for Test 1 was a single rock taken from the 700 pound sulfide sample. It was crushed to minus 1 1/2" in the laboratory jaw crusher, slurried to 50 percent solids, lime was added to raise the pH to 11.7, 10 pounds of cyanide per ton was added, and the sample was agitated for 48 hours in a rolling bottle.

The samples for tests 2 through 5 were prepared by splitting the 700 pound sample in half. One half was crushed to minus 1 1/2 inch. Five thousand gram samples for tests 2 and 4 were split out. The remaining 1 1/2 inch ore was split in half and one half was crushed to minus 3/4 inch. Five thousand gram samples for tests 3 and 5 were split out.

Tests 2 and 3 were assay screen analyses for ore samples crushed to minus 1 1/2 and minus 3/4 inches, respectively. Test 4 and 5 were cyanide leach amenability tests for ore samples crushed to minus 1 1/2 and minus 3/4 inches. The samples were slurried to 50 percent solids, lime was added to raise the pH to 11.2, 10 lbs NaCN per ton of solution was added and the samples were agitated in rolling bottles for 48 hours. Assay screen analyses were made on the leach residues.

We appreciate the opportunity to work with you. If you have any questions, please contact us.

Very truly yours,
DAWSON METALLURGICAL LABORATORIES, INC.

Philip Thompson

Philip Thompson,
Vice President

PT-cac

DAWSON
METALLURGICAL
LABORATORIES, INC.

P. O. Box 7685
5217 Major Street
Murray, Utah 84107
Phone: 801-262-0922

PROJECT NO. P-1045-L
DATE 9/3/85
BY MT
Sulfide Ore

TEST NO. 1 NAME Lacanca Gold

Cyanide Amenability @ 1 1/2 inch - 48 hours - Assay Screen on Residue

PRODUCT	Weight	Percent Weight	ASSAY Au	ASSAY Ag	UNITS Au	UNITS Ag	DISTRIBUTION Au	DISTRIBUTION Ag
Leach Residue								
+1"	2319.0	46.5	0.041	<.05	0.0191	0.0093	49.2	21.7
+3/4"	998.0	20.0	0.041	0.06	0.0082	0.0120	21.1	28.0
+1/2"	478.0	9.6	0.033	0.09	0.0032	0.0086	8.3	20.0
+1/4"	328.0	6.6	0.026	0.04	0.0017	0.0026	4.4	6.1
-1/4"	864.0	17.3	0.038	0.06	0.0066	0.0104	17.0	24.2
	4987.0	100.0	0.039	0.04	0.0388	0.0429	100.0	100.0
Leach Solution	4943.0		0.010	0.05	.0494	.2472	20.25	55.34
Leach Residue	4987		0.039	0.04	.1945	.1995	79.75	44.66
Head (calc)			0.049	0.09	.2439	.4467	100.00	100.00

OPERATION							
	Leach			Leach			
TIME				11:20	4:50		48 hr
REAGENTS - LBS PER TON				Start			Off
Ore (-1 1/2 inch)	5000						
Water	5000						
Lime grams		3.0				0.7	
NaCN grams				25.0			
Lime Titration, lb/T Soln							0.1
NaCN Titration, lb/T Soln							7.6
Lime Consumed, lb/T Ore							1.1
NaCN Consumed, lb/T Ore							2.5
MACHINE							
R.P.M.							
pH	8.5		11.7		10.2		10.3
% SOLIDS							
TEMPERATURE							

GRINDING PRODUCT

MESH	%	%
+10		
+14		
+20		
+28		
+35		
+48		
+65		
+100		
+150		
+200		
+325		
-325		

REMARKS:

* Split -1/4" in half, hold 1/2 to send, 1/2 for assay

DML
DAWSON METALLURGICAL LABORATORIES, INC.
P. O. Box 7685
5217 Major Street
Murray, Utah 84107
Phone: 801-262-0922

PROJECT NO. P-1045-L
DATE 9/12/85
BY MT
-1 1/2" Assay Screen

TEST NO. 2 NAME Lacana

Assay Screen Head Sample Crushed to -1 1/2"

PRODUCT	Weight	PERCENT WEIGHT	ASSAY					UNITS			DISTRIBUTION				
			Au	Ag				Au	Ag			Au	Ag		
+1"	855.0	16.0	0.028	0.20				0.0045	0.0321			9.7	7.5		
+3/4"	1444.0	27.1	0.030	0.12				0.0081	0.0325			17.5	7.6		
+1/2"	943.0	17.7	0.045	0.52				0.0080	0.0920			17.3	21.6		
+1/4"	730.0	13.7	0.040	0.70				0.0055	0.0959			11.9	22.5		
+35 Mesh	969.0	18.2	0.052	0.41				0.0095	0.0745			20.6	17.5		
-35 Mesh	389.0	7.3	0.145	1.35				0.0106	0.0986			23.0	23.3		
Head (calc)	5330.0	100.0	0.046	0.43				0.0462	0.4256			100.0	100.0		

												GRINDING PRODUCT	
OPERATION													
TIME													
REAGENTS - LBS PER TON													
											MESH	%	%
											+10		
											+14		
											+20		
											+28		
											+35		
											+48		
											+65		
											+100		
											+150		
MACHINE												+200		
R.P.M.												+325		
pH												-325		
% SOLIDS													
TEMPERATURE													

REMARKS:

DAWSON METALLURGICAL LABORATORIES, INC.
P. O. Box 7685
5217 Major Street
Murray, Utah 84107
Phone. 801-262-0922

PROJECT NO. P-1045-L
DATE 9/12/85
BY MT
-3/4" Assay Screen

TEST NO. 3 NAME Lacana
Assay Screen Head Sample Crushed to - 3/4"

PRODUCT	Weight	PERCENT WEIGHT	ASSAY					UNITS			DISTRIBUTION				
			Au	Ag				Au	Ag			Au	Ag		
+1/2"	1055.0	21.0	0.060	0.19				0.0126	0.040			22.62	7.96		
+1/4"	1665.0	33.2	0.036	0.40				0.0119	0.1328			21.36	26.41		
+35	1741.0	34.7	0.051	0.61				0.0177	0.2116			31.78	42.08		
-35	555.0	11.1	0.122	1.07				0.0135	0.1184			24.24	23.55		
Head (calc)	5016	100.0	0.056	0.50				0.0557	0.5028			100.0	100.0		

OPERATION	GRINDING PRODUCT		
TIME			
REAGENTS - LBS PER TON			
	MESH	%	%
	+10		
	+14		
	+20		
	+28		
	+35		
	+48		
	+65		
	+100		
	+150		
MACHINE	+200		
R.P.M	+325		
pH	-325		
% SOLIDS			
TEMPERATURE			

REMARKS:

DAWSON METALLURGICAL LABORATORIES, INC.

P. O. Box 7685
5217 Major Street
Murray, Utah 84107
Phone. 801-262-0922

PROJECT NO. P-1045-L
DATE 9/13/85
BY MT
-1 1/2" Crush

TEST NO. 4 NAME Lacana

48 hour NaCN Leach with 10 lbs/ton NaCN Solution. Assay Screen Leach Residue

PRODUCT	Weight	PERCENT WEIGHT	ASSAY					UNITS			DISTRIBUTION				
Leach Residue			Au	Ag				Au	Ag			Au	Ag		
+1"	840.0	16.8	0.031	0.06				0.0052	0.0101			14.90	4.71		
+3/4"	1079.0	21.5	0.035	0.08				0.0075	0.0172			21.49	8.02		
+1/2"	801.0	16.0	0.024	0.46				0.0038	0.0735			10.89	34.27		
+1/4	579.0	11.6	0.024	0.48				0.0028	0.0555			8.02	25.87		
+35 Mesh	839.0	16.7	0.030	0.16				0.0050	0.0268			14.33	12.49		
-35 Mesh	873.0	17.4	0.061	0.18				0.0106	0.0314			30.37	14.64		
Total Weight	5011.0	100.0	0.035	0.21				0.0349	.2145			100.0	100.0		
Leach Residue	5011.0		0.035	0.21				0.1754	1.0523			81.05	42.32		
Leach Solution	5123.0		0.008	0.28				0.0410	1.4344			18.95	57.68		
Head (calc)	5011		0.043	0.5				0.2164	2.4867			100.0	100.0		

OPERATION				Leach		Off							GRINDING PRODUCT	
TIME				1:20		48hrs								
REAGENTS - LBS PER TON				Start										
												MESH	%	%
-1 1/2 Ore	5000											+10		
Water	5000											+14		
Lime, grams		4.0			1							+20		
NaCN, grams				25.0								+28		
NaCN Titration, lb/T Soln						5.3						+35		
CaO Titration, lb/T Soln						< .1						+48		
NaCN Consumed, lb/T Ore						4.6						+65		
Lime Consumed, lb/T Ore						2.0						+100		
												+150		
MACHINE												+200		
RPM												+325		
pH	6.8		11.2			10.5						-325		
% SOLIDS														
TEMPERATURE														

REMARKS:

DAWSON METALLURGICAL LABORATORIES, INC.
P. O. Box 7685
5217 Major Street
Murray, Utah 84107
Phone. 801-262-0922

PROJECT NO. P-1045-L
DATE 9/13/85
BY MT
-3/4" Crush

TEST NO. 5 NAME Lacana

48 hour NaCN Leach with 10 lbs/ton NaCN Solution. Assay screen leach residue.

PRODUCT	Weight	PERCENT WEIGHT	ASSAY Au	ASSAY Ag	UNITS Au	UNITS Ag	DISTRIBUTION Au	DISTRIBUTION Ag
Leach Residue								
+3/4"	0.0	0.0						
-3/4 +1/2"	756.0	15.1	0.032	.06	0.0048	0.0091	12.09	7.27
-1/8 +1/4"	1436.0	28.7	0.035	.08	0.0100	0.0230	25.19	18.37
-1/4 +35 Mesh	1676.0	33.5	0.035	.19	0.0117	0.0636	29.47	50.80
-35 Mesh	1136.0	22.7	0.058	.13	0.0132	0.0295	33.25	23.56
Total Weight	5004	100.0	0.04	0.12	0.0397	0.1252	100.0	100.0
Leach Residue	5004		0.04	0.12	0.2002	0.6005	66.71	27.31
Leach Solution	4995		0.02	0.32	0.0999	1.5984	33.29	72.69
Head (calc)	5004		0.06	0.44	0.3001	2.1989	100.0	100.0

OPERATION				Leach		Off
TIME				1:30		48 hrs
REAGENTS - LBS PER TON				Start		
-3/4" Ore	5000					
Water	5000					
Lime, gram		4.0			1	
NaCN, gram				25.0		
NaCN Titration, lb/t Soln						4.6
CaO Titration, lb/t Soln						< .1
NaCN Consumed, lb/t Ore						5.4
Lime Consumed, lb/t Ore						2.0
MACHINE						
R P.M						
pH	6.8		11.2			10.3
% SOLIDS						
TEMPERATURE						

GRINDING PRODUCT

MESH	%	%
+10		
+14		
+20		
+28		
+35		
+48		
+65		
+100		
+150		
+200		
+325		
-325		

REMARKS:

ASSAY REPORT SHEET

ASSAY LAB, INC.
1376 W. 8040 So. Unit #4
West Jordan, Utah 84C84

Date Received ____________ Date Reported 9/12/85

Client Dawson Metallurgical Labs

Sample Identification		Oz / Ton Au	Oz / Ton Ag	Remarks
P-1045C				* Ounces per ton of 2000 lbs.
Lacana				
Leach Res.				
	-¼	.036	.05	
		.040	.06	
	-½+¼	.028	.05	
		.025	.04	
	+1"	.040	< .05	
		.042	< .05	
	-1"+3/4	.042	.07	
		.040	.05	
	-3/4+½	.032	.11	
		.034	.08	
Leach Solution				
Test #1		.011	.05	
		.010	.05	

ASSAY REPORT SHEET

ASSAY LAB, INC.
1376 W. 8040 So. Unit #4
West Jordan, Utah 84084

Date Received ____________ Date Reported 9/19/85

Client Dawson Metallurgical Lab

Sample Identification	Oz / Ton Au	Oz / Ton Ag	Remarks
P-1045-L			* Ounces per ton of 2000 lbs.
Lacana			
Test #2			
Assay Screen			
+1"	.028	.19	
	.029	.20	
+3/4	.029	.13	
	.030	.10	
+½	.045	.55	
	.045	.50	
+¼	.040	.68	
	.039	.71	
35 mesh	.050	.40	
	.054	.42	
-35 mesh	To Follow		
Test #3			
+½	.062	.21	
	.058	.17	
+¼	.037	.44	
	.034	.38	
+35 mesh	.052	.64	
	.050	.58	
-35 mesh	.124	1.08	
	.122	1.06	
Test #4			
Leach Res.			
+1"	.030	.07	
	.032	.05	
-1"+3/4	.035	.10	
	.039	.06	
-3/4+½	.026	.45	
	.023	.48	
-½+¼	.023	.46	
	.024	.50	
-¼+35	.031	.16	
	.030	.17	
-35 mesh	.063	.19	
	.059	.18	
Test #5 -3/4+½	.034	<.05	
	.031	.06	

ASSAY REPORT SHEET

ASSAY LAB, INC.
1376 W. 8040 So. Unit #4
West Jordan, Utah 84084

Date Received ____________ Date Reported ____________

Client ____________________

Sample Identification	Oz / Ton Au	Oz / Ton Ag	Remarks
$-\frac{1}{2}+\frac{1}{4}$	.036	.10	* Ounces per ton of 2000 lbs.
	.034	.07	
$-\frac{1}{4}+35$ mesh	.035	.16	
	.035	.22	
-35 mesh	.056	.14	
	.060	.12	
Leach Soln.			
Test #4	.008	.29	
	.008	.28	
Test #5	.019	.31	
	.020	.33	

Ronald Bianchi

ASSAY REPORT SHEET

ASSAY LAB, INC.
1376 W. 8040 So. Unit #4
West Jordan, Utah 84084

Date Received ____________ Date Reported 9/20/85

Client Dawson Metallurgical

Sample Identification	Oz / Ton Au	Oz / Ton Ag	Remarks
P-1045L Lacana Test #2			* Ounces per ton of 2000 lbs.
-35 mesh	.144	1.25	
	.146	1.45	

Ronald Bianchi

P-1000

This X-Copy for
Ron Graichen

CYPRUS METALLURGICAL PROCESSES CORPORATION

TUCSON, ARIZONA

FILE NUMBER: 823-42-5001

SUBJECT: CYANIDATION OF GILT EDGE ORE

AUTHOR: Jerry E. Dobson

REPORT DATE: May 12, 1982

May 12, 1982

SUMMARY

The cyanidation of Gilt Edge ore in an agitated leaching operation may be expected to yield about 75 to 80% of the gold content or 0.046 oz/ton on a weighted average basis. This assumes a grinding to about 70% -200 mesh; marginal increase in yield to the 85% range might be expected with the samples ground to 100% -200 mesh. The consumption of sodium cyanide, of course, increases with the fineness of grind reaching an average of 72 lb/oz Au in our most finely ground samples. Lime demand, on the other hand, showed only modest increases during the same experiments; about 100-110 lb/oz Au is required.

The brief examination of concentrates revealed that flotation may easily recover ca. 85% of the gold value and that upon cyanidation approximately 84% of this is recoverable. The net yield then is about 72% with the advantage of about 90% less bulk to be treated. The average grade of concentrate treated was 14.9 ppm Au yielding 0.37 oz Au/Ton of concentrate. Cyanide consumption was approximately the same as the unconcentrated ore at 30-40 lb/oz Au; lime usage decreased sharply to about 13 lb/oz Au.

The leaching of the roasted concentrate gave significantly greater recovery of 97% of the gold as expected since the occlusion of particles in the pyrite matrix is probably responsible for their inactivity.

Roasting in conjunction with flotation will recover about 82% of the gold value with reduced cyanide and grinding costs. These factors in addition to the size reduction of concentrate handling facilities may justify more thorough evaluation of the flotation recovery limits.

CYANIDATION OF GILT EDGE ORE

INTRODUCTION

Samples of gold ore from several diamond drill hole cores and composites of the Gilt Edge prospect were received for cyanidation testwork. The furnished samples ranged in gold content from 0.7 to 7.7 ppm gold and from 1.5 to 20 ppm silver content. The leaching tests were directed toward the treatment of agitated ore pulps although some flotation concentrates as well as roasted concentrates were leached. The latter effort resulted from a spate of erroneous assays which led us to conclude mistakenly that the gold value was quite refractory.

EXPERIMENTAL

Sample Preparations

The various ore samples were reduced from the as received condition to about -14 mesh using jaw and roller crushing. Samples of the crushed core specimens were split out for head assays, test samples and a reserve supply.

Further size reduction was carried out in a laboratory steel ball mill or in the instance of concentrates, which were small samples, by hand in a mortar and pestle.

Samples which were roasted were treated in an oven operating at between 600 and 625°C.

CYANIDATION OF GILT EDGE ORE

Analytical

The metal values in both ore residue and solution was monitored by atomic absorption spectroscopy. In the case of gold some difficulties arose which resulted in poor accountability and delayed production of believable extraction data. Some liquor samples, perhaps related to the sulfide content of the ores, seem to undergo a reductive loss of part of their gold content with time. Delays in assaying as short as one day may be serious in the matter of gold accountability under these circumstances.

LEACHING PROCEDURES

Cyanide leachings of ores and concentrates were carried out using the rolled bottle method of agitation. Using untreated ores, 200g samples were employed per test whereas flotation concentrates or roasted concentrate samples were leached on a 20g scale. All leachings were performed on 45% solids in aqueous NaCN slurry.

The concentrations of the metal values developed in the leaching solution were monitored as a function of time. Similarily the consumption of lime and sodium cyanide during the dissolution was measured. The test samples were leached a minimum of 24 hours and occasionally longer.

Records of quantities were kept entirely by weight necessitating only that a thorough washing of solid be achieved to have accuracies within the limits of the assay precision.

CYANIDATION OF GILT EDGE ORE

Flotation

Each ore sample was subjected to a rough flotation expected to recover its pyritic fractions. Samples were ground to an intermediate size in seven minutes of grinding, the pH adjusted to the range from 7.5 to 8.5 using Na_2CO_3 and brought to ca. 30% solids. A total dose of 0.1 lb/ton i-amylxanthate was added over a ten minute flotation time and MIBC was used as needed for froth. Head, concentrate, and tails assays indicated typical recoveries of about >80% of the gold content and 60% of silver.

RESULTS

Table I presents the referencing identification for the Amoco Minerals Company's sample designation and the letter identification assigned for convenience by Cymet. For quick reference the overall performance of the leaching of gold from each of the various samples under the several conditions employed in this study is also reported. The degree of grinding is designated by the series A thru E for each sample in order of increasing grind time. The split at 200 mesh was measured and is keyed at the bottom of the table. The quantity of gold developed in the cyanide leachate for these various conditions is reported in the fourth column in ounces per ton of ore leached and was based upon the quantity of gold detected in solutions after the cyanidation reaction. As to leaching efficiency, the calculated head derived from product assays was used to determine the percentage reported in the fifth column. Finally, the sixth column of Table 1 records the consumption of NaCN per ton of ore.

The remaining tables detail the individual leachings including not only the gold results but silver and copper extractions as well.

Table II sets out the cyanidation efficiencies which were found for several samples of Gilt Edge ore when cyanidation was tested on coarsely ground materials. Because of the sometimes difficulty in accountability, perhaps because of coarse gold, we report two extraction values in this numerical tabulation. The first is the extraction based upon the average gold content of the head samples; the second extraction column is based upon the level of gold found in that particular sample's leached products, i.e. a calculated head basis. The third column lists the mass balance across the leaching process from average head composition to leached tailings and liquor levels. Similarily, Columns 4, 5, and 6 report the corresponding results calculated for the silver content of the ore which, though generally low, were also monitored. Strong cyanide extraction, 0.2%, of the gold from these rather coarsely crushed samples established a base with which to compare other conditions. This was the most coarsely crushed of the samples measuring about 22% -200 mesh fraction. The recovery was inadequate, averaging only 51%.

A somewhat different format is employed in Tables III through VI and VIII to take advantage of a computer printout. It is self explanatory in large, but contains more information. The columns under the heading Assays give the nead, leach liquor, tails and calculated head values. Extractions are reported based both upon calculated head values, which are preferred, as well as head values which are included for the sake of confidence as well as a measure by which to gauge the balances.

Table III summarizes the leaching results for an intermediate grind of the ores. A typical screen analysis in this sample set yielded 45% - 200 mesh fraction. This resulted in a substantially better degree of leaching than was given by the coarsely ground samples in Table II. The extractions averaged 74% based upon calculated head values and 74% as a gold weighted average as well.

Table IV and V are the result of yet finer grinding at 65% and 70% -200 mesh respectively. This spacing is closer than planned but the data of both are included to increase the data base. The only difference, other than the marginal size distribution change, was that the NaCN level of Table V (70% -200 mesh) was reduced to 0.05% to verify the usual lack of effect of CN^- concentration upon leaching kinetics in the ranges being employed. As may be seen from the individual tests and the weighed averages presented in Table I the lowered cyanide level may have had some effect, but this is primarily due to depletion between samplings rather than a bona fide kinetic rate effect, i.e. the reaction time was truncated by reagent consumption. The extractions in Table IV and V calculated as a straight average were 79 and 71% respectively. Calculated weighted average based upon contained gold values were 83 and 74%.

CYANIDATION OF GILT EDGE ORE

Table VI furnishes the data of the cyanidation behavior of the most finely ground set of samples, corresponding to test E of the summary Table I. These leachings attempted to remove particle size from consideration as a limiting factor in dissolution. All were subjected to twenty minutes grinding in the steel mill and reported >98% as a -200 mesh fraction, in fact they were >95% -325 mesh. The background cyanide level was restored to 0.2% NaCN in order to handle any increase in copper and acid activity resulting from enhanced oxidation at this very fine state of subdivision. The average extraction was 76% as a straight average and 80% as a weighted average. This seems to be biased by two very poor performances by samples D and K in this experiment.

Table VII assembles the data concerning the small investigation of concentrating the ore. This is included for completeness, however, the reason for its existence was based upon some assay difficulties which, when resolved, faded along with the need of concentration. As mentioned earlier, no optimization of flotation recovery was attempted, merely a rougher concentration in order to attain sufficient concentrate for testing. Any assessment of concentration or concentration and roasting as possible processing steps would require additional evaluation. In this table we report the overview of the results obtained in concentrating and concentrate leaching the gold from each ore sample. A composite ore sample was also processed through each operation. The final entry in the table gives the weighed average gold extraction from the concentrates D thru L.

CYANIDATION OF GILT EDGE ORE

Table VIII provides the detailed test by test data from the leaching of the concentrates and the composite concentrate. Also included here is the leaching behavior of the composite concentrate after four hours roasting at 600°C which reduced the sulfur content from >30 to >2%. The format of this table parallels those given as III through VI.

The figure presented gives the extraction curve for gold as a function of the fineness of grind. The general feature is obvious and expected in the indication of higher extraction resulting from increasing particle subdivision. A principle feature would appear to be the rapid increase in leachability as the quantity of -200 mesh material increased from about 20% to 45%; further grinding did not dramatically affect recovery (see weighted average extraction for A thru E grinds in Table I). The latter two data points for grinds D and E may, however, as mentioned before, somewhat underestimate extraction. If so, the flattening of the curve should not be as promounced as portrayed in the Figure. The recoverable upper limit of gold from this ore may thus approach 90% under the conditions employed here.

CYANIDATION OF GILT EDGE ORE

TABLE I

Sample Referencing and Overview of Results

Sample: Cymet Letter	Sample: AMOCO Designation		Gold OPT	Recovery %	Leaching Behavior: NaCN Consumed #T
D	GLE Composite	A			
	DDH#21 50'-444"	B	0.028	71	2.6
	X-21 TP mixed	C	0.028	71	2.8
		D	0.025	63	3.4
		E	0.018	43	4.5
E	GLE Composite	A	0.059	61	2.0
	DDH#22 450'-740'	B	0.053	79	0.4
		C	0.064	85	0.4
	X-22 ITP-sulfide	D	0.068	77	0.6
	mixed + oxide	E	0.060	91	3.3
F	GLE Composite	A	0.014	41	0.5
	DDH#22 80'-180'	B	0.021	55	0.8
	& 320'-450'	C	0.025	68	1.2
	TP-sulfide	D	0.021	55	3.0
	X-22	E	0.028	71	4.6
G	81 DDH-6	A	0.122	57	0.6
	610'-680'	B	0.132	76	0
	Sample "A"	C	0.160	90	0.3
	X-6 TP-sulfide	D	0.146	82	0.4
		C	0.160	96	2.5
H	81 DDH-6	A	0.029	46	0.3
	680'-775'	B	0.046	76	2.4
	Sample "B"	C	0.043	83	0.3
	X-6 TP-sulfide	D	0.046	73	1.6
		E	0.053	95	1.8
I	81 DDH-16	A			
	100'-200'	B	0.021	88	1.7
	Sample "A"	C	0.021	88	1.9
		D	0.021	88	1.0
	X-16 ITP + Rhy	E	0.021	88	1.8
	oxide				

CYANIDATION OF GILT EDGE ORE

TABLE I (con't)

Sample Referencing and Overview of Results

Sample: Cymet Letter	Sample: AMOCO Designation		Gold Recovery OPT	%	Leaching Behavior: NaCN Consumed #T
J	81 DDH-16 240'-295' Sample "A" X-16 ITP - oxide	A			
		B	0.032	85	1.5
		C	0.046	89	0.3
		D	0.036	86	1.8
		E	0.036	92	2.6
K	81 DDH-17 9' - 196' X-17 ITP oxide + sulfide	A			
		B	0.021	71	3.3
		C	0.021	71	2.6
		D	0.014	49	3.6
		E	0.007	26	4.6
L	81 DDH-17 196'391' Sample "A" X-17 ITP - sulfide	A			
		B	0.025	68	1.1
		C	0.025	68	1.2
		D	0.036	71	1.1
		E	0.028	83	3.4
D thru L - Weighted Average of all Samples		A	0.054	51%*	Only D,E,F&G
		B	0.042	74%	
		C	0.048	83%	
		D	0.046	74%	
		E	0.044	80%	

Grind A 22% -200 mesh
B 45% -200 mesh
C 65% -200 mesh
D 70% -200 mesh
E 98% -200 mesh

TABLE II

(Gilt Edge Ore Cyanidation)

Sample	% Au Ext H	-CH	% Bal	% Ag Ext H	-CH	% Bal	#/T NaCN	#/T CaO
E	60	61	98	58	21	282	2	8
F	32	41	78	54	56	96	0.5	8
G	54	56	96	35	35	100	0.6	6.7
H	45	46	98	16	16	101	0.3	6.7

Fire Assay Au

Sample	% Au Ext H	-CH	% Bal
E	100	71	141
F	55	39	72
G	73	70	104
H	56	72	77

Typical screen analysis: 36.1%-65; 22.2%-200

NaCN: 0.2%
Time: 24 hours
H: Based on head assay
CH: Based on calculated head assay.

40 TO 50%-200 mesh,0.2% NaCN,5#/T CaO,24 hrs.

TABLE III

Test ID	*	Assays ppm As	ppm Au	ppm Cu	% Extraction As	Au	Cu	% Balance As	Au	Cu	Reagen consumpt. #/T CN	# C
D	H	3.6	1.6	295.	58.	61.	64.	99.	86.	113.		
	L	1.7	0.8	154.								
	T	1.5	0.4	146.								
	C	3.6	1.4	334.	58.	71.	56.				2.6	4
	OPT				0.060	0.028	0.380					
E	H	2.1	3.4	114.	46.	54.	30.	118.	69.	98.		
	L	0.8	1.5	28.								
	T	1.5	0.5	78.								
	C	2.5	2.3	112.	39.	79.	30.				0.4	4
	OPT				0.028	0.053	0.070					
F	H	6.0	1.5	233.	53.	49.	57.	498.	89.	102.		
	L	2.6	0.6	109.								
	T	26.7	0.6	104.								
	C	29.9	1.3	237.	11.	55.	56.				0.8	4.
	OPT				0.093	0.021	0.270					
G	H	6.1	7.7	183.	34.	59.	11.	509.	77.	112.		
	L	1.7	3.7	16.								
	T	29.0	1.4	186.								
	C	31.1	5.9	206.	7.	76.	9.				-0.0	4.
	OPT				0.060	0.132	0.040					
H	H	5.9	2.1	298.	17.	76.	9.	95.	99.	116.		
	L	0.8	1.3	22.								
	T	4.6	0.5	318.								
	C	5.6	2.1	345.	18.	76.	8.				2.4	4.
	OPT				0.028	0.046	0.050					
I	H	1.5	1.0	49.	49.	73.	25.	155.	83.	147.		
	L	0.6	0.6	10.								
	T	1.6	0.1	60.								
	C	2.3	0.8	72.	31.	88.	17.				1.7	4.
	OPT				0.021	0.021	0.020					

* H=head L=liquor T=tail C=calculated head OPT=tr.oz./Ton
@=lbs./Ton

40 TO 50%-200 mesh,0.2% NaCN,5#/T CaO,24 hrs.

Test ID	*	Assays ppm Ag	ppm Au	ppm Cu	% Extraction Ag	Au	Cu	% Balance Ag	Au	Cu	Reagent consumpti #/T CN	#/ C
J	H	2.7	1.6	152.	63.	69.	9.	137.	81.	81.		
	L	1.4	0.9	11.								
	T	2.0	0.2	109.								
	C	3.7	1.3	122.	46.	85.	11.				1.5	4.
	OPT				0.050	0.032	0.03@					
K	H	19.9	1.4	736.	61.	52.	63.	98.	81.	108.		
	L	9.9	0.6	383.								
	T	7.4	0.4	329.								
	C	19.5	1.1	796.	62.	65.	59.				3.3	4.
	OPT				0.352	0.021	0.93@					
L	H	4.6	1.6	255.	48.	53.	68.	2863.	78.	105.		
	L	1.8	0.7	142.								
	T	129.5	0.4	94.								
	C	131.7	1.3	267.	2.	68.	65.				1.1	4.
	OPT				0.064	0.025	0.35@					

* H=head L=liquor T=tail C=calculated head. OPT=tr.oz./Ton
@=lbs./Ton

approx.65%-200 mesh,0.2% NaCN,5#/T CaO,24 hrs.

TABLE IV

Test ID	*	Assays ppm Ag	ppm Au	ppm Cu	% Extraction Ag	Au	Cu	% Balance Ag	Au	Cu	Reagent consumpti $/T CN	#/ Ca
D	H	3.6	1.6	295.	61.	61.	67.	83.	86.	101.		
	L	1.8	0.8	163.								
	T	0.8	0.4	99.								
	C	3.0	1.4	298.	73.	71.	67.				2.8	5.
	OPT				0.064	0.028	0.40@					
E	H	2.1	3.4	114.	46.	65.	45.	65.	76.	127.		
	L	0.8	1.8	42.								
	T	0.8	0.4	93.								
	C	1.8	2.6	144.	55.	85.	36.				0.4	4.
	OPT				0.028	0.064	0.10@					
F	H	6.0	1.5	233.	53.	57.	64.	81.	84.	108.		
	L	2.6	0.7	123.								
	T	1.7	0.4	102.								
	C	4.9	1.3	252.	65.	68.	60.				1.2	4.
	OPT				0.093	0.025	0.30@					
G	H	6.1	7.7	183.	36.	71.	23.	88.	79.	113.		
	L	1.8	4.5	35.								
	T	3.2	0.6	165.								
	C	5.4	6.1	208.	41.	90.	21.				0.3	4.
	OPT				0.064	0.160	0.07@					
H	H	5.9	2.1	298.	21.	70.	8.	129.	84.	107.		
	L	1.0	1.2	20.								
	T	6.4	0.3	295.								
	C	7.6	1.8	319.	16.	63.	8.				0.3	4.
	OPT				0.036	0.043	0.05@					
I	H	1.5	1.0	49.	73.	73.	32.	153.	83.	171.		
	L	0.9	0.6	13.								
	T	1.2	0.1	68.								
	C	2.3	0.8	84.	48.	88.	19.				1.9	4.
	OPT				0.032	0.021	0.03@					

* H=head L=liquor T=tail C=calculated head OPT=tr.oz./Ton
@=lbs./Ton

approx.65%-200 mesh,0.2% NaCN,5#/T CaO,24 hrs.

Test ID	*	Assays ppm As	Assays ppm Au	Assays ppm Cu	% Extraction As	% Extraction Au	% Extraction Cu	% Balance As	% Balance Au	% Balance Cu	Reagent consumpt. $/T CN	$/ C
J	H	2.7	1.6	152.	68.	99.	10.	90.	112.	82.		
	L	1.5	1.3	12.								
	T	0.6	0.2	110.								
	C	2.4	1.8	125.	75.	89.	12.				0.3	4
	OPT				0.053	0.046	0.03@					
K	H	19.9	1.4	736.	68.	52.	69.	115.	74.	111.		
	L	11.1	0.6	413.								
	T	9.3	0.3	309.								
	C	22.8	1.0	819.	59.	71.	62.				2.6	4
	OPT				0.395	0.021	1.02@					
L	H	4.6	1.6	255.	56.	53.	70.	110.	78.	101.		
	L	2.1	0.7	147.								
	T	2.5	0.4	78.								
	C	5.1	1.3	257.	51.	68.	70.				1.2	4
	OPT				0.075	0.025	0.36@					

* H=head L=liquor T=tail C=calculated head OPT=tr.oz./Ton
@=lbs./Ton

Approx. 70% -200 mesh, 0.05% NaCN, 24 hrs.

TABLE V

Test ID	*	Assays ppm Ag	ppm Au	ppm Cu	% Extraction Ag	Au	Cu	% Balance Ag	Au	Cu	Reagent consumpt. #/T NaCN	# C
D	H	3.6	1.6	295.	64.	54.	60.	126.	86.	97.		
	L	1.9	0.7	146.								
	T	2.3	0.5	109.								
	C	4.6	1.4	287.	50.	63.	62.				3.4	5
	OPT				0.065	0.025	0.360					
E	H	2.1	3.4	114.	64.	68.	41.	187.	89.	113.		
	L	1.1	1.9	38.								
	T	2.6	0.7	83.								
	C	3.9	3.0	129.	34.	77.	36.				0.6	5
	OPT				0.039	0.068	0.090					
F	H	6.0	1.5	233.	59.	48.	69.	98.	87.	97.		
	L	2.9	0.6	131.								
	T	2.3	0.6	66.								
	C	5.8	1.3	226.	61.	55.	71.				3.0	5
	OPT				0.103	0.021	0.320					
G	H	6.1	7.7	183.	42.	65.	13.	102.	79.	102.		
	L	2.1	4.1	19.								
	T	3.7	1.1	163.								
	C	6.3	6.1	186.	41.	82.	12.				0.4	5
	OPT				0.075	0.146	0.050					
H	H	5.9	2.1	298.	23.	76.	16.	96.	104.	102.		
	L	1.1	1.3	40.								
	T	4.3	0.6	254.								
	C	5.6	2.2	303.	24.	73.	16.				1.6	6
	OPT				0.039	0.045	0.100					
I	H	1.5	1.0	49.	75.	76.	17.	137.	87.	124.		
	L	0.9	0.6	7.								
	T	0.9	0.1	52.								
	C	2.0	0.8	61.	55.	88.	14.				1.0	[illegible]
	OPT				0.032	0.031	0.020					

* H=head L=liquor T=tail C=calculated head OPT=tr.oz./Ton

@=lbs./Ton

Approx. 70% -200 mesh, 0.05% NaCN, 24 hrs.

Test ID	*	Assays ppm As	Assays ppm Au	Assays ppm Cu	% Extraction As	% Extraction Au	% Extraction Cu	% Balance As	% Balance Au	% Balance Cu	Reagent consumpt #/T CN	Reagent consumpt #/T C
J	H	2.7	1.6	152.	72.	76.	14.	112.	88.	84.		
	L	1.6	1.0	18.								
	T	1.1	0.2	105.								
	C	3.1	1.4	127.	64.	86.	17.				1.8	6
	OPT				0.057	0.036	0.040					
K	H	19.9	1.4	736.	75.	36.	70.	108.	73.	104.		
	L	12.3	0.4	422.								
	T	6.4	0.5	254.								
	C	21.4	1.0	769.	70.	49.	67.				3.6	5
	OPT				0.438	0.014	1.030					
L	H	4.6	1.6	255.	67.	75.	69.	111.	106.	104.		
	L	2.5	1.0	144.								
	T	2.0	0.5	89.								
	C	5.1	1.7	265.	60.	71.	66.				1.1	5
	OPT				0.089	0.036	0.350					

* H=head L=liquor T=tail C=calculated head OPT=tr.oz./Ton
@=lbs./Ton

[illegible]

TABLE VI

Test ID	*	Assays ppm As	Assays ppm Au	Assays ppm Cu	% Extraction As	% Extraction Au	% Extraction Cu	% Balance As	% Balance Au	% Balance Cu	Reagent consumpti[on] $/T CN	$/ Ca
D	H	3.6	1.6	295.	47.	39.	71.	100.	90.	117.		
	L	1.4	0.3	172.								
	T	1.9	0.8	136.								
	C	3.6	1.4	346.	47.	43.	61.				4.5	4.
	OPT				0.030	0.012	0.420					
E	H	2.1	3.4	114.	64.	61.	19.	116.	67.	127.		
	L	1.1	1.7	18.								
	T	1.1	0.2	123.								
	C	2.4	2.3	145.	55.	91.	15.				3.3	5.
	OPT				0.039	0.060	0.048					
F	H	6.0	1.5	233.	39.	64.	87.	88.	90.	115.		
	L	1.9	0.8	167.								
	T	2.9	0.4	64.								
	C	5.2	1.4	268.	44.	71.	76.				4.6	5.
	OPT				0.063	0.028	0.410					
G	H	6.1	7.7	183.	42.	71.	26.	101.	74.	110.		
	L	2.1	4.5	39.								
	T	3.6	0.2	153.								
	C	6.2	5.7	201.	42.	96.	24.				2.5	5.
	OPT				0.075	0.160	0.100					
H	H	5.9	2.1	298.	21.	87.	16.	100.	92.	104.		
	L	1.0	1.5	40.								
	T	4.7	0.1	261.								
	C	5.9	1.9	310.	21.	95.	16.				1.8	5.
	OPT				0.036	0.053	0.100					
J	H	1.5	1.0	49.	92.	76.	32.	112.	87.	203.		
	L	1.1	0.6	21.								
	T	0.3	0.1	75.								
	C	1.6	0.8	101.	82.	88.	25.				1.8	4.
	OPT				0.039	0.021	0.050					

* H=head L=liquor T=tail C=calculated head OPT=tr.oz./Ton
P=lbs./Ton

[illegible]: 86% -325 mesh, .2% NaCN, 24 hrs.

Test ID	*	Assays ppm As	ppm Au	ppm Cu	% Extraction As	Au	Cu	% Balance As	Au	Cu	Reagent consumpt $/T CN	# C
J	H	2.7	1.6	152.	94.	76.	31.	131.	82.	109.		
	L	2.1	1.0	38.								
	T	1.0	0.1	120.								
	C	3.6	1.3	166.	72.	92.	28.				2.6	4
	OPT				0.075	0.036	0.09@					
K	H	19.9	1.4	736.	10.	18.	55.	98.	70.	100.		
	L	1.6	0.2	331.								
	T	17.6	0.7	330.								
	C	19.6	0.9	734.	10.	26.	55.				4.6	4
	OPT				0.057	0.007	0.81@					
L	H	4.6	1.6	255.	72.	60.	80.	107.	72.	107.		
	L	2.7	0.8	168.								
	T	1.6	0.2	67.								
	C	4.9	1.2	272.	67.	83.	75.				3.4	4
	OPT				0.096	0.028	0.41@					

* H-head L=liquor T=tail C=calculated head. OPT=tr.oz./Ton
@=lbs./Ton

CYANIDATION OF GILT EDGE ORE

TABLE VII

Flotation

Sample	Float Recovery (%) Au	Float Recovery (%) Ag	Grade (ppm) Au	Grade (ppm) Ag	Leachability (%) Au	Leachability (%) Ag	#/TNaCN
D	77	79	9.0	22.1	73	55	11.1
E	82	87	18.8	22.5	84	52	10.8
F	88	71	7.5	20.6	68	59	11.0
G	79	86	55.3	43.8	88	39	10.5
H	95	72	21.8	51.9	81	69	10.7
oxide I	45	-	8.0	18.3	95	79	12.3
oxide J	51	39	13.3	17.9	97	84	11.8
K	75	57	5.5	61.8	58	48	11.6
L	79	36	7.5	18.6	73	63	11.2
Gold Weighted Average D thru L					84	59	11.2
Composite Conc.	85	70	15.8	30.3	83	59	11.0
Roasted Compo Conc			19.9	35.4	97	23	4.7

TABLE VIII

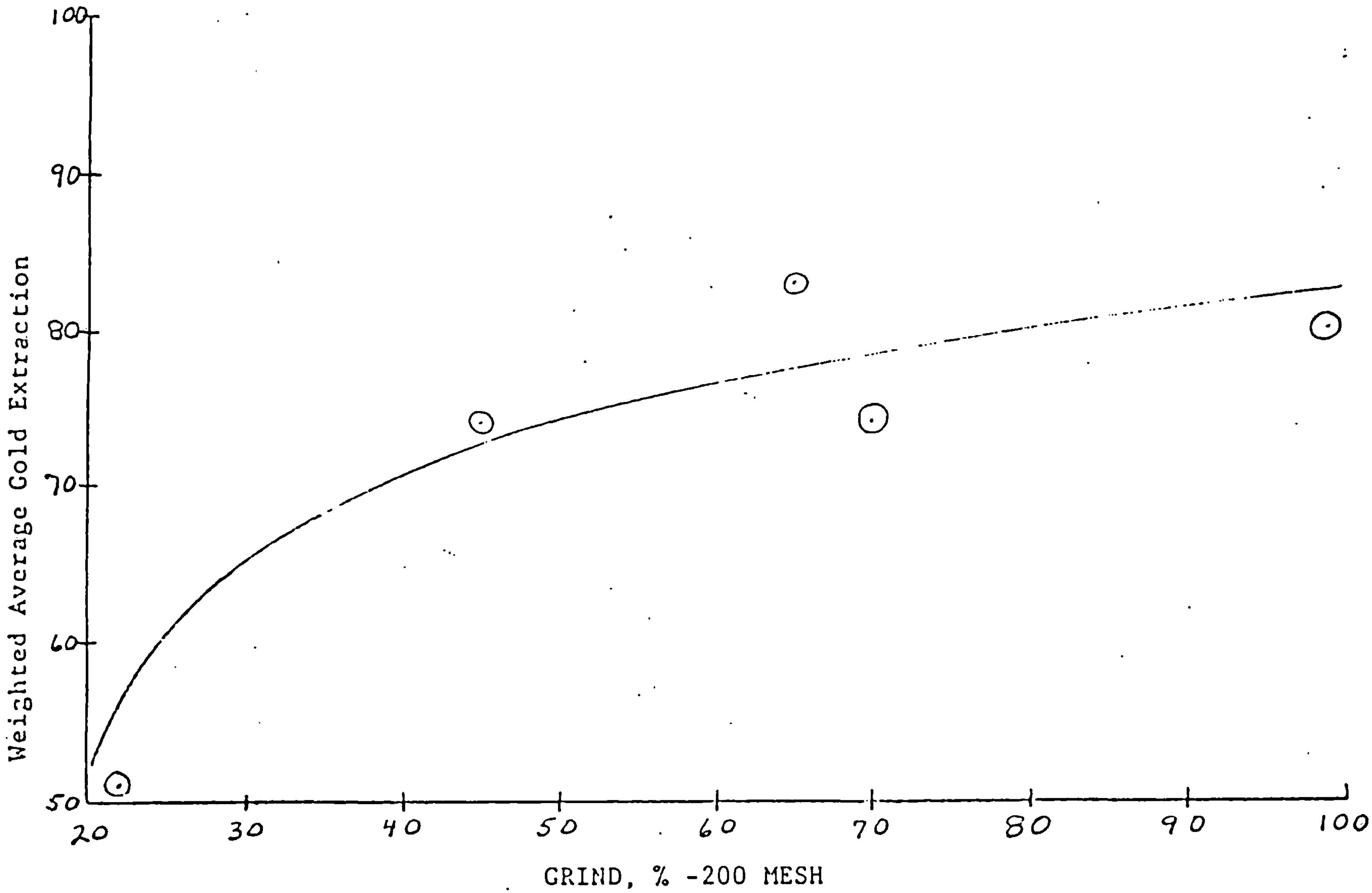
Test ID	*	Assays ppm As	ppm Au	ppm Cu	% Extraction As	Au	Cu	% Balance As	Au	Cu	Reagent consumpt. $/T CN	$/ C
D	H	22.1	9.0	1334.	44.	68.	74.	79.	93.	93.		
	L	7.9	5.0	809.								
	T	7.8	2.3	250.								
	C	17.4	8.4	1237.	55.	73.	80.				11.1	4
	OPT				0.281	0.178	1.97@					
E	H	22.5	18.8	735.	31.	83.	42.	61.	98.	105.		
	L	5.8	12.8	251.								
	T	6.6	2.9	496.								
	C	13.7	18.5	802.	52.	84.	36.				10.8	4
	OPT				0.206	0.455	0.61@					
F	H	20.6	7.1	904.	64.	67.	70.	108.	98.	104.		
	L	10.8	3.9	521.								
	T	9.1	2.2	309.								
	C	22.3	7.0	945.	59.	68.	67.				11.0	4
	OPT				0.384	0.139	1.27@					
G	H	43.8	49.1	1862.	38.	78.	9.	99.	88.	106.		
	L	13.7	31.5	134.								
	T	26.5	5.0	1816.								
	C	43.2	43.4	1979.	39.	88.	8.				10.5	4
	OPT				0.487	1.121	0.33@					
H	H	51.9	21.8	3914.	194.	86.	5.	282.	106.	98.		
	L	82.5	15.3	148.								
	T	45.9	4.4	3661.								
	C	146.6	23.1	3842.	69.	81.	5.				10.7	4
	OPT				2.936	0.544	0.36@					
I	H	18.3	8.0	356.	23.	89.	44.	29.	94.	339.		
	L	3.4	5.8	129.								
	T	1.1	0.4	1049.								
	C	5.3	7.5	1207.	79.	95.	13.				12.3	6
	OPT				0.121	0.207	0.32@					

* H=head L=liquor T=tail C=calculated head OPT=tr.oz./Ton
@=lbs./Ton

Test ID	*	Assays ppm Ag	ppm Au	ppm Cu	% Extraction Ag	Au	Cu	% Balance Ag	Au	Cu	Reagent consumpt #/T CN	#/ C
J	H	17.9	13.3	407.	51.	103.	35.	60.	106.	227.		
	L	7.4	11.2	115.								
	T	1.7	0.4	784.								
	C	10.7	14.1	924.	84.	97.	15.				11.8	5.
	OPT				0.264	0.399	0.28@					
K	H	61.8	5.5	2423.	54.	51.	80.	113.	67.	97.		
	L	27.3	2.3	1584.								
	T	36.6	2.0	419.								
	C	69.9	4.8	2351.	48.	58.	82.				11.6	4.
	OPT				0.971	0.082	3.86@					
L	H	18.6	7.5	1362.	65.	73.	83.	103.	100.	98.		
	L	9.9	4.5	730.								
	T	7.0	2.0	203.								
	C	19.1	7.5	1338.	63.	73.	85.				11.2	4.
	OPT				0.352	0.160	2.27@					
Cmp	H	30.3	15.1	1646.	60.	69.	51.	102.	83.	69.		
	L	14.9	8.6	686.								
	T	12.7	2.1	626.								
	C	30.9	12.6	1463.	59.	83.	57.				11.0	4.
	OPT				0.530	0.306	1.67@					

* H=head L=liquor T=tail C=calculated head OPT=tr.oz./Ton
@=lbs./Ton

. IGURE: GOLD EXTRACTION VS GRIND



LACANA MINING INC.

MEMORANDUM October 29, 1983

TO: PAUL DIRCKSEN

FROM: ROD MACLEOD

SUBJECT: GILT EDGE PROJECT, Comparison of "Main GILT EDGE" ROTARY HOLES

INTRODUCTION

Of the 41 reverse circulation rotary holes drilled this summer, the last 5 were drilled within the "Main Gilt Edge area". These 5 holes (RGE-37 through 41) were drilled in known areas of relatively high Au mineralization and with the exception of RGE-39, were drilled to a depth of 305 feet. RGE-39 was terminated prematurely at 205 feet when it intersected a stope (?) in the old Rattlesnake Jack workings. For each of these 5 holes, all of the cuttings were collected from each 5 foot interval so they can be used for metallurgical testing. In addition, a split from each 5 foot interval was obtained for assay; serving as a "check" against assay data from nearby Cyprus (Amoco) rotary holes. The geology was logged from the assay splits, giving particular care in noting oxide vs. sulphide.

ASSAY COMPARISON

On the accompanying pages, the assays from several Cyprus rotary holes located around LACANA rotary holes have been tabulated with the LACANA holes. A location map has also been included for reference. Collar elevations are given in parenthesis under the hole numbers and points of equal elevations are marked with an asterisk (*) in the assay tabulations.

As an initial means of comparison, all of the assays from each hole were averaged. These averages are given at the end of each hole. LACANA assays generally compare quite well with the Cyprus assays. In some cases a few ten's of feet of high Au mineralization carried the average for the entire hole (e.g., GLE-69, page 2 of tabulations). In addition, a comparison of assay numbers at the same or very nearly the same elevation generally indicates the intervals of relatively high Au intercepts in one hole can be correlated to a similar high Au intercept in another hole.

Some of the Cyprus holes around RGE-37, 39, and 41 have considerably lower Au mineralization than the LACANA RGE holes. The exact cause of this lower Au mineralization is uncertain, but some possibilities are: (1) inadequate rock preparation for mineralizing fluids; (2) increasing distance from the "central and southern Gilt Edge stocks"; and (3) host rock type. An example of the importance of host rock type (and rock preparation) was found in RGE-40 between 217 feet and 249 feet where a dike or sill(?) of "sanidine rhyolite porphyry" was intersected. Except for the assays that included the upper and lower contacts, Au mineralization was nil or very low at best, in the rhyolite. By comparison, the trachyte (Amoco's fine-grained rhyolite), which was the rock type in the rest of the hole, had relatively high Au values. From my mapping and core logs, the central stock of sanidine rhyolite porphyry postdates the trachyte porphyry and Au mineralization is frequently very low in the central stock. It seems quite probable that a similar explanation can hold for the upper 230 feet of GLE-69 (page 2 of tabulations).

Finally, it is my hope that when the core drilling is completed in the "Main Gilt Edge area" that I can go back through the rotary chips (and core) from the Cyprus holes with high Au mineralization and log the oxide vs. sulphide to get a much more accurate distribution of oxidation. The means by which oxide vs. sulphide data was documented prior to LACANA, seems tenuous at best.

Cyprus Conventional GLE-199 (5446')	LGI R.C. RGE-37 (5504')		Interval	
.008	.026	.008	0-5	MIXED OXIDE/ SULPHIDE
		.044	5-10	
.010	NS	NS	10-15	
		NS	15-20	
.022	±.028	NS	20-25	
		.028	25-30	
.016	.023	.014	30-35	
		.032	35-40	
* .030	.018	.020	40-45	
		.016	45-50	
.050	.012	.012	50-55	
		.012	55-60	
.038	.010	.010	60-65	
		.010	65-70	
.010	.017	.016	70-75	
		.018	75-80	
.010	.015	.012	80-85	
		.018	85-90	
.036	.018	.020	90-95	
		.018	95-100	
.008	.035	.052	100-105	
		.018 *	105-110	
.028	.022	.028	110-115	
		.016	115-120	
.040	.015	.016	120-125	
		.014	125-130	SULPHIDE
.042	.015	.010	130-135	
		.020	135-140	
.040	.006	.010	140-145	
		.002	145-150	
.018	.018	.012	150-155	
		.024	155-160	
.018	.035	.030	160-165	
		.040	165-170	
.016	.059	.062	170-175	
		.056	175-180	
.024	.082	.070	180-185	
		.094	185-190	
.018	.068	.054	190-195	
		.082	195-200	
.014	.036	.042	200-205	
		.030	205-210	
.028	.028	.030	210-215	
		.026	215-220	
.022	.053	.054	220-225	
		.052	225-230	
.026	.056	.054	230-235	
		.058	235-240	
.014	.066	.048	240-245	
		.084	245-250	
Ave.=.023	.070	.082	250-255	
		.058	255-260	
	.067	.056	260-265	
		.078	265-270	
	.058	.086	270-275	
		.030	275-280	
	.038	.040	280-285	
		.036	285-290	
	.043	.044	290-295	
		.042	295-300	
		.064	300-305	
		Ave.=.036		

Cyprus Conven. GLE-23 (5616')	LGI R.C. RCE-39 (5621'')		Interval	
.011(3-10')	.011	.006	0-5	OXIDE
		.016	5-10	
.014	.011	.010	10-15	
		.012	15-20	
.014	.013	.014	20-25	
		.012	25-30	
.020	.019	.014	30-35	
		.024	35-40	
.011	.020	.016	40-45	
		.024	45-50	
.019	.057	.056	50-55	
		.058	55-60	
.014	.015	.020	60-65	
		.010	65-70	
.014	.012	.016	70-75	
		.008	75-80	
.017	.025	.026	80-85	
		.024	85-90	
.024	.020	.020	90-95	
		.020	95-100	
.042	.020	.016	100-105	
		.024	105-110	
.012	.008	.006	110-115	
		.010	115-120	
.023	.025	.014 *	120-125	
		.036	125-130	
.041	.014	.014	130-135	
		.014	135-140	
.041	.026	.012	140-145	
		.040	145-150	
.023	.018	.022	150-155	
		.014	155-160	MIXED OXIDE/ SULPHIDE
.099	.058	.070	160-165	
		.046	165-170	
.038	.038	.040	170-175	
		.036	175-180	
.032	.031	.044	180-185	
		.018	185-190	
.080	.021	.016	190-195	
		.026	195-200	
Ave.=.029		.024	200-205	
		Ave.=.023		

	Cyprus Convention GLE-25 (5611')	GLE-47 (5625')	GLE-24 (5619')	GLE-218 (5630')	LGI & C. RGE-40 (5620')		
0-10	.024(5-10')	.003	.020(3-10')	.024	.048	.032 .064	0-5 5-10
10-20	.014	.010	.010	.064	.023	.018 .028	10-15 15-20
20-30	.033	.045	.010	.106	.012	.012 .012	20-25 25-30
30-40	.010	.002	.009	.048	.033	.034 .032	30-35 35-40
40-50	.011	.002	.010	.032	.077	.144 .010	40-45 45-50
50-60	.031	.008	.020	.016	.009	.010 .008	50-55 55-60
60-70	.050	.020	.017	.034	.019	.008 .030	60-65 65-70
70-80	.095	.029	.035	.040	.017	.012 .022	70-75 75-80
80-90	.234	.012	.017	.036	.035	.020 .050	80-85 85-90
90-100	.087	.027	.011	.244	.022	.032 .012	90-95 95-100
100-110	* .014	.017	.023	.034	.011	.014 .008	100-105 105-110
110-120	.021	.029	* .027	Ave.=.062	.042	.008 .076	110-115 115-120
120-130	.036	* .002	.009		.031	.016 .046	120-125 125-130
130-140	.038	.002	.005	*	.025	.018 .032	*130-135 135-140
140-150	.086	.013	.009		.108	.032 .184	140-145 145-150
150-160	.165	.020	.009		.119	.162 .076	150-155 155-160
160-170	.042	.002	.008		.033	.012 .054	160-165 165-170
170-180	.024	.012	.013		.057	.048 .066	170-175 175-180
180-190	.071	.001	.014		.026	.020 .032	180-185 185-190
190-200	.062	.050	.027		.040	.060 .020	190-195 195-200
200-210	Ave.=.057	.036	.032		.027	.024 .030	200-205 205-210
210-220		< .001	.048		.020	.024 .016	210-215 215-220
220-230		< .001	.026		.004	.004 .004	220-225 225-230
230-240		< .001	.022		.008	nil .008	230-235 235-240
240-250		< .001	.016		.102	.120 .048	240-245 245-250
250-260		.002	.020		.026	.026 .026	250-255 255-260
260-270		.120	.090		.022	.018 .026	260-265 265-270
270-280		.120	.031		.068	.012 .124	270-275 275-280
280-290		.099	.048		.027	.012 .042	280-285 285-290
290-300		.159	.165		.064	.082 .046	290-295 295-300
300-310		.210	Ave.=.027			.114	300-305
						Ave.≃.040	
310-320		.162					
320-330		.120					
330-340		.081					
340-350		.081					
		Ave.≃.043					

OXIDE

* = 5500' elevation

	Cyprus Conventional						LGI R.C		
	GLE-43 (5560')	GLE-151 (5547')	GLE-15 (5578')	GLE-16 (5582')	GLE-45 (5580')	GLE-46 (5600')	RGE-41 (5572')		
0-10	.003	.008	NS(0-15')	NS(0-15')	.007	.026	.021	.022 .020	0-5 5-10
10-20	< .001	Tr.	.004	.002	.007	.019	.042	.018 .066	10-15 15-20
20-30	.003	.008	.008	.016	.003	.014	.091	.088 .094	20-25 25-30
30-40	.002	.008	.004	.002	.001	.006	.170	.124 .216	30-35 35-40
40-50	<.001	* .003	.005	.002	.001	.021	.059	.028 .090	40-45 45-50
50-60	*<.001	.010	.008	.004	<.001	.030	.046	.072 .020	50-55 55-60
60-70	<.001	.020	.021	.056	.001	.072	.044	.046 .042	60-65 65-70
70-80	<.001	.049	* .034	.078 *	*<.001	.090	.041	.032 .050	* 70-75 75-80
80-90	<.001	.115	.080	.012	.001	.108	.063	.040 .086	80-85 85-90
90-100	.015	.015	.111	.010	.001	* .216	.066	.096 .036	90-95 95-100
100-110	.001	.018	.069	.007	.028	.360	.084	.042 .126	100-105 105-110
110-120	.007	Tr.	.024	.012	.014	.564	.046	.034 .058	110-115 115-120
120-130	.002	Tr.	.022	.020	.024	.816	.038	.054 .022	120-125 125-130
130-140	.003	.015	.050	.026	.001	.510	.026	.030 .022	130-135 135-140
140-150	.001	.010	.026	.062	.066	.276	.039	.038 .040	140-145 145-150
150-160	.002	.005	.032	.031	.036	.372	.024	.026 .022	150-155 155-160
160-170	<.001	.010	.038	.023	.024	.150	.042	.044 .040	160-165 165-170
170-180	.002	.010	.011	.083	.014	.132	.050	.024 .076	170-175 175-180
180-190	.001	.010	.012	.064	.002	.570	.175	.114 .236	180-185 185-190
190-200	<.001	.012	.009	.060	.002	.430	.074	.088 .060	190-195 195-200
200-210	.001	Tr.	.009	.156	.005	.009	.047	.064 .030	200-205 205-210
210-220	.003	Tr.	.010	.083	.009	.033	.064	.054 .074	210-215 215-220
220-230	.004	Tr.	.008	.026	.014	.150	.012	.018 .008	220-225 225-230
230-240	.004	.016	.010 Ave.=.026	.048	.026 Ave.=.012	.020 Ave.=.212	.013	.010 .016	230-235 235-240
240-250	.007	.010		.020 Ave.=.038			.013	.012 .014	240-245 245-250
250-260	.004 Ave.≃.003	.014					.017	.014 .020	250-255 255-260
260-270		.016					.019	.022 .016	260-265 265-270
270-280		.011					.012	.012 .012	270-275 275-280
280-290		.020					.019	.010 .028	280-285 285-290
290-300		.019					.014	.016 .012	290-295 295-300
300-310		.008 Ave.≃.015						.022 Ave.=.061	300-305

* = 5500' elevation

RGE-41

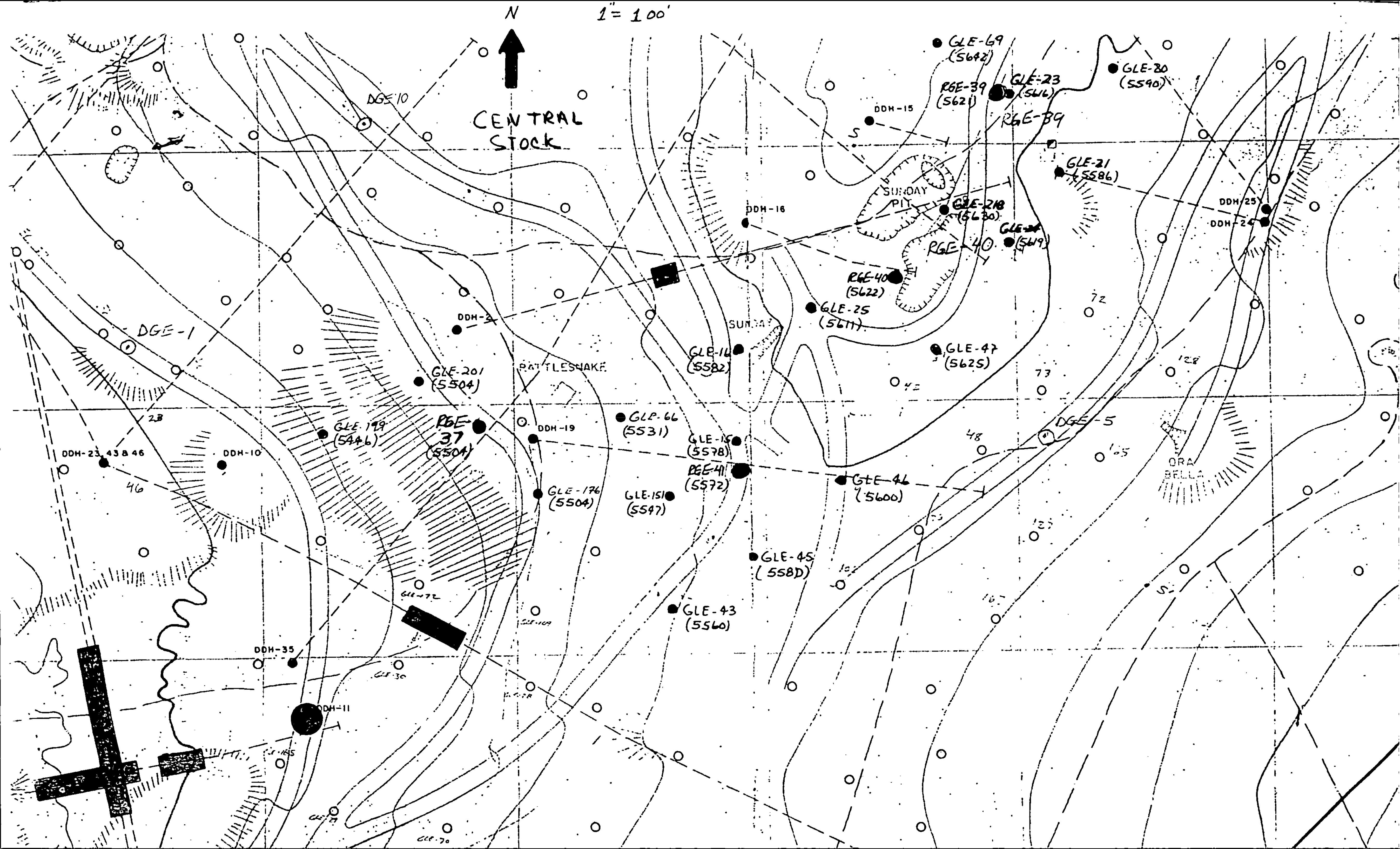
0-135' -- OXIDE
135-175' -- MIXED OXIDE/SULPHIDE
175-205' -- OXIDE
205-305' -- MIXED OXIDE/SULPHIDE

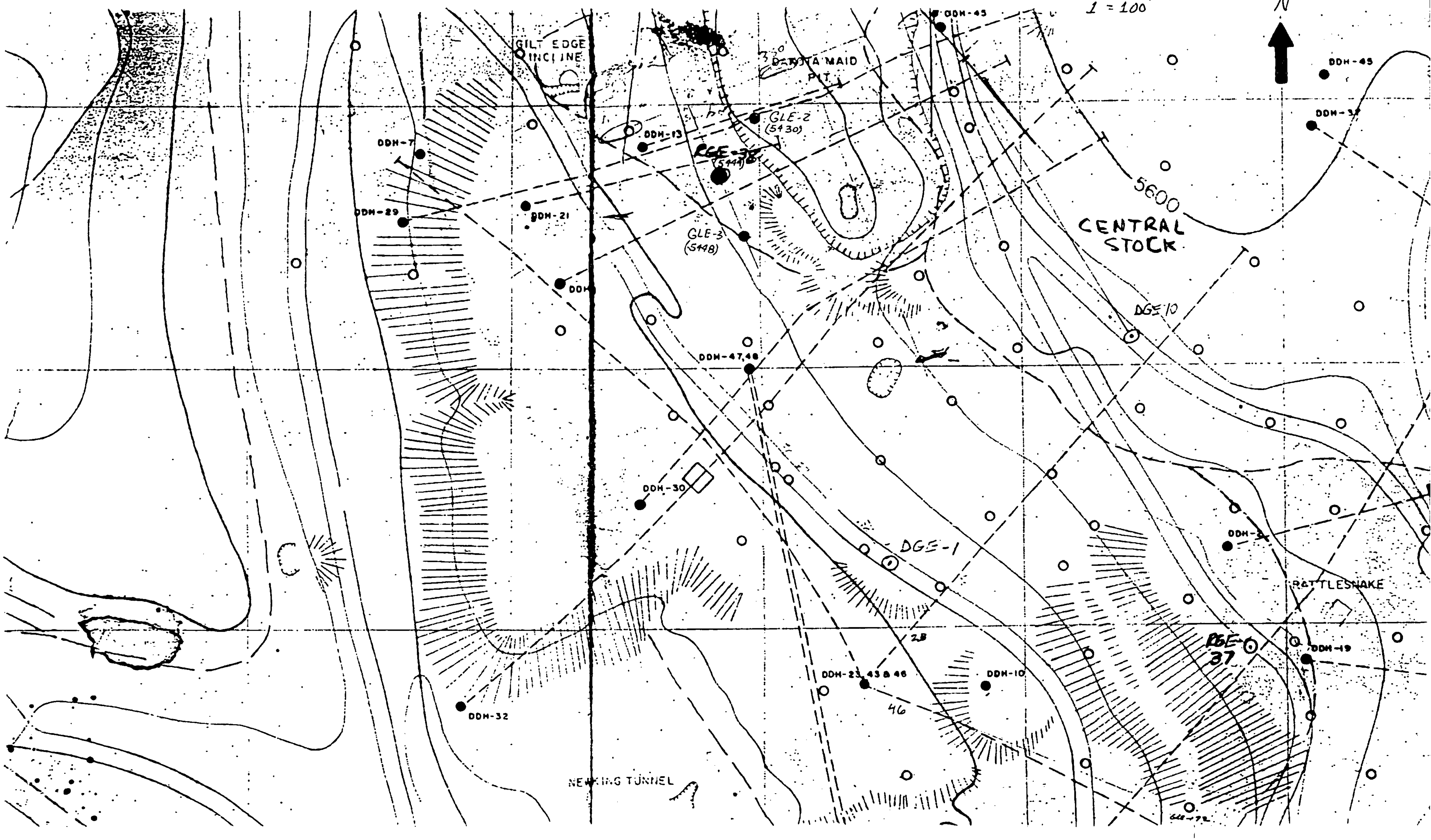
~~Cyprus~~ Cyprus Conv.

LGI -5 R.C.

Interval	GLE-2 (5430')	GLE-3 (5448')	RGE-38 (5449')		Interval	Zone
0-10	.285(2-10')	.030	.039	.022	0-5	OXIDE
				.056	5-10	
10-20	.180	.036	.086	.092	10-15	
				.080	15-20	
20-30	* .114	.039	.056	.046	20-25	
				.066	25-30	
30-40	.065	.020	.036	.052	30-35	
				.020	35-40	
40-50	.102	* .020	.043	.034	40-45	
				.052	45-50	
50-60	.060	.026	.037	.048	50-55	
				.026 *	55-60	0.044
60-70	.123	.013	.042	.040	60-65	
				.044	65-70	
70-80	.099	.018	±.026	NS	70-75	
				.026	75-80	
80-90	.047	.038	.048	.030	80-85	
				.066	85-90	
90-100	.053	.035	.048	.064	90-95	
				.032	95-100	
100-110	.051	.114	.036	.038	100-105	
				.034	105-110	
110-120	.042	.038	.034	.042	110-115	
				.026	115-120	MIXED OXIDE/ SULPHIDE
120-130	.027	.047	.064	.048	120-125	
				.080	125-130	
130-140	.044	.077	.083	.070	130-135	
				.096	135-140	SULPHIDE
140-150	.023	.023	.039	.032	140-145	
				.046	145-150	
150-160	.048	.044	.037	.036	150-155	
				.038	155-160	
160-170	.045	.027	.035	.040	160-165	
				.030	165-170	
170-180	.020	.044	.040	.032	170-175	
				.048	175-180	
180-190	.023	.026	.067	.118	180-185	
				.016	185-190	
190-200	.020	.024	.013	.014	190-195	
	Ave.=.074			.012	195-200	
200-210		.012	.053	.032	200-205	
				.074	205-210	
210-220		.008	.090	.032	210-215	
				.148	215-220	
220-230		.015	.069	.080	220-225	
				.058	225-230	
230-240		.023	.041	.036	230-235	
				.046	235-240	
240-250		.027	.027	.024	240-245	
		Ave.=.033		.030	245-250	
			.029	.040	250-255	
				.018	255-260	
			.042	.034	260-265	
				.050	265-270	
			.109	.148	270-275	MIXED OXIDE/ SULPHIDE
				.070	275-280	
			.106	.152	280-285	
				.060	285-290	
			.046	.048	290-295	
				.044	295-300	
				.024	300-305	
				Ave.=.051		

* = 5400' elevation


N
1" = 100'
CENTRAL STOCK
DGE 10
DGE-1
DGE-5
GLE-69 (5642)
GLE-23 (5616)
RGE-39 (5621)
RGE-39
GLE-20 (5590)
DDH-15
GLE-21 (5586)
SUNDAY PIT
GLE-218 (5630)
DDH-16
DDH-25
DDH-24
RGE-40
RGE-40 (5622)
GLE-25 (5611)
GLE-47 (5625)
GLE-16 (5582)
RATTLESNAKE
GLE-201 (5504)
RGE-37 (5504)
GLE-199 (5446)
GLE-66 (5531)
DDH-19
GLE-15 (5578)
RGE-41 (5572)
GLE-46 (5600)
DDH-2
DDH-23, 43 & 46
DDH-10
GLE-176 (5504)
GLE-151 (5597)
GLE-45 (558D)
GLE-43 (5560)
ORA BELLA
DDH-35
DDH-11



1" = 100'
N
GILT EDGE INCLINE
DAKOTA MAID PIT
DDH-45
DDH-45
DDH-37
DDH-7
DDH-13
GLE-2 (5430)
RGE-38 (5444)
DDH-29
DDH-21
GLE-3 (5448)
5600
CENTRAL STOCK
DGE 10
DDH-47,48
DDH-30
DGE-1
DDH-32
DDH-23, 43 & 46
46
DDH-10
RATTLESNAKE
RGE-37
DDH-19
NEW KING TUNNEL